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PHASE I
CHARACTERIZATION OF NON-FERROUS SLAGS FROM CANADIAN SOURCES

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JOHN J. EMERY

TROW LTD.

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43 BAYWOOD ROAD
REXDALE, ONTARIO
M9V 3Y8
(416) 749-1290

SUMMARY
PHASE I
CHARACTERIZATION OF NON-FERROUS SLAGS FROM CANADIAN SOURCES

This report gives the findings of Phase I of the study on the characterization of non-ferrous slags from Canadian sources for cementitious applications, including two comparison foreign non-ferrous slags. The overall study is concerned with the technical aspects of using ground non-ferrous slags as a partial replacement for Portland cement in cemented mine backfill and/or concrete. Phase I of the study involved: developing a sampling program; visiting Canadian nickel, copper and lead smelters; grinding non-ferrous slags to specific Blaine finenesses with comparison to cement clinker and pelletized iron blast furnace slag; chemical and compositional characterization; and pozzolanic and hydraulic index testing of eleven non-ferrous slags.

I appears that three factors contribute to decreased non-ferrous slag grindability - increasing vitrification, decreasing SiO_2 content and decreasing Fe_2O_3 content - with degree of vitrification predominating. While optical microscopy and qualitative X-ray diffraction were used to determine glass contents, a better method such as the CANMET image analysis method should be adopted. The ASTM C989 index (50 percent Portland cement replacement and 28 days curing at 23°C) for 4000 Blaine was found to be the best indicator of potential cementitious behaviour. Strength development generally increases with increasing vitrification. The strength development results are considered promising, and the Phase II study on concrete mix designs and testing should be completed.

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INTRODUCTION

Canada, as a large world supplier of nickel, copper and lead, has a number of non-ferrous smelters that produce significant quantities of by-product non-ferrous slags. Except for rail ballast and granular fill applications, little of this non-ferrous slag is utilized, leading to the disposal of substantial quantities and large accummulations of non-ferrous slag in smelter areas. Research has been conducted by several Canadian companies (Inco, Falconbridge, Kidd Creek Mines, St. Lawrence Cement and Canada Cement Lafarge) on the use of ground non-ferrous slag as a partial replacement for Portland cement in cemented mine backfill. However, this has not reached the regular use stage that is current practice at Australia's Mount Isa Mine (ground, granulated copper reverberatory slag replacing fifty percent of the Portland cement), and there are logistical problems given the relatively slow rate of strength development as ground non-ferrous slags tend to act as pozzolans.

For satisfactory cementitious performance, it is fairly clear from the technical literature on cemented mine backfill that the non-ferrous slag should be: vitrified (rapidly quenched through granulation, noting that ambient cooled may be suitable in some cases as some vitrification still occurs); finely ground (higher Blaine fineness than Portland cement, noting that greater grinding energy/cost required in terms of both grinding resistance and higher Blaine); and replacement level limited for satisfactory reactivity (both rate of compressive strength development and level achieved). Given the mining and construction requirements for Portland cement in smelter areas of Canada, and their typical long distances from cement production areas, the partial replacement of Portland cement with ground non-ferrous slag in cemented mine backfill (i.e. similar to Mount Isa Mine) and/or concrete (not current practice) would result in substantial cost savings and reduced slag disposal problems.

Evaluating the technical feasibility of the partial replacement of Portland cement with typical ground Canadian nickel, copper and lead slags (with foreign sources for comparison purposes) is the overall objective of this two phase study that complements research in the CANMET Construction Materials Section. More specifically, Phase I reported herein was concerned with: developing a sampling program for non-ferrous slags (Canadian and foreign); visiting Canadian nickel, copper and lead smelters to organize slag sampling and to become familiar with smelter processes; grinding the non-ferrous slags to specific Blaine finenesses (2000, 3000, 4000 and 5000), with comparison to cement clinker and pelletized iron blast furnace slag, and determination of grinding energy requirements; chemical and compositional characterization of the non-ferrous slags; and pozzolanic and hydraulic index testing of ASTM C109 mortar cubes (mainly ASTM C595 and C989 procedures) incorporating the ground non-ferrous slags at various Portland cement replacement levels, including sodium hydroxide activation and replacement of Portland cement with cement kiln dust. The subsequent Phase II will involve concrete mix designs and testing for six ground non-ferrous slags selected on the basis of the Phase I findings.

As a preliminary to this Phase I study, the overall current and potential applications for non-ferrous slags were reviewed based on the technical literature and discussions with industry representatives, as summarized in Appendix A and listed in Table 1 with several key references. The available information on ground non-ferrous slags as a pozzolanic/hydraulic cementitious material indicates that while its reactivity is slow, that it can be used as a partial replacement for Portland cement. This replacement level, while maintaining satisfactory strength development, must be determined for specific non-ferrous slags (i.e. source, type and cooling procedure). These aspects are addressed in following sections of the Phase I study, mainly through grinding, chemical and compositional characterization of eleven representative non-ferrous slags, followed by mortar cube compressive strength testing at various Portland cement replacement levels, temperatures and curing times. The overall performance of the resulting cemented fill or concrete, as determined by appropriate procedures, will form the subsequent Phase II study.

NON-FERROUS SLAGS AND TAILINGS SAMPLES

A glossary for all of the Canadian and foreign slags and tailings (for Phase II) obtained for the study, along with a source, type and cooling description, is given in Table 2. Reference 7 (report references given in

TABLE 1

APPLICATIONS FOR NON-FERROUS SLAGS AND MAJOR REFERENCES

(Also see Appendix A)

A. CURRENT^{C} AND POTENTIAL^{P} APPLICATIONS

(Storage and/or Landfilling not Included)

1. Nickel Slag
   - i) granulated:
     - a) mining - cemented mine backfill^{P} (prime potential cementitious application), mine backfill^{C}.
     - b) transportation - anti-skid material^{C}, base stabilization^{P} (cementitious application).
     - c) other - blended cement^{P} (cementitious application), sand blasting media^{C}, waste treatment and sludge solidification^{P} (cementitious application), brick making^{P}.
   - ii) ambient cooled:
     - a) transportation - railway ballast^{C} (major Canadian use), highway and mining road subbase and base construction^{C}, erosion protection^{C}, embankment construction^{C}, stabilization of soft soils^{C}, asphaltic concrete aggregate^{P}.
     - b) buildings and dams - foundation support^{C}, underfloor fill^{C}, starter dams for tailings storage^{C}, stabilizing berms for dams^{C}, filter zones for dams^{C}, erosion protection^{C}, concrete aggregate^{P}.
     - c) other - ambient cooled may have some cementitious applications as per granulated (i.e. ambient cooled has some vitrification).
2. Copper Slag (Applications similar to those for nickel slag.)
   - i) granulated:
     - a) mining - cemented mine backfill^{C} (prime cementitious application). Remainder as per nickel slag.
   - ii) ambient cooled:
     As per nickel slag with addition of slag wool manufacture^{P}.

TABLE 1 Cont.
APPLICATIONS FOR NON-FERROUS SLAG AND MAJOR REFERENCES
A. CURRENT^C AND POTENTIAL^P APPLICATIONS, Cont.

3. Lead and Lead-Zinc
   - i) granulated:
     - a) mining - cemented mine backfill^P (prime potential cementitious application).

     Remainder as per nickel slag and copper slag.

   - ii) ambient cooled:

     As per nickel slag and copper slag with addition of ceramics manufacture^P and pigment manufacture^C.

NOTE

Leachate and environmental constraints must be checked for most applications, particularly for lead and lead-zinc slags.

APPLICATIONS FOR NON-FERROUS SLAG AND MAJOR REFERENCES

B. MAJOR REFERENCES

1. OECD, Use of Waste Materials and By-products in Road Construction, Organization for Economic Co-operation and Development, Paris, 1977.

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3. Gutt, W. and Nixon, P.J., Use of Waste Materials in the Construction Industry - Analysis of the Rilem Symposium by Correspondence, Matériaux et Constructions, Vol. 12, No. 70, 1979.

4. Winer, A.A. and Wang, S.B., Mineral Insulation - A Critical Study, CANMET Report 80-11E, EMR, Ottawa, 1980.

5. Collings, R.K. and Wang, S.S.B., Mineral Waste Resources of Canada Report No. 8 - Non-ferrous Metallurgical Wastes, CANMET Report 81-ME, EMR, Ottawa, 1981.

6. Emery, J.J., Slag Utilization in Pavement Construction, ASTM STP 774, Extending Aggregate Resources, ASTM, Philadelphia, 1982.

7. Douglas, E., Characterization of Non-ferrous slags of Canadian Origin - Report on Field Trips Related to the Slag Sampling Program, CANMET Report No. MSL-INT 83-4 (FT), EMR, Ottawa, 1983.

8. Douglas, E., Canadian Ferrous and Non-ferrous Slags for Resource and Energy Conservation, ERP/MSL Report No. 83-61 (TR), EMR, Ottawa, 1983.

Also see Appendix A and References Cited in the Papers Listed Above and in Appendix A.

TABLE 2

GLOSSARY FOR SAMPLES

SLAGS

- BDLG - Belledune, N.B., lead blast furnace slag, fresh, granulated. (Brunswick Mining and Smelting Corporation)
- ICNA - Sudbury, Ont., nickel reverberatory slag, fresh, ambient cooled. (INCO Metals Company)
- ICNR - Sudbury, Ont., nickel slag (ICNA) remelted in laboratory, vitrified by quenching.(INCO Metals Company)
- KCCG - Timmins, Ont., copper smelter slag, fresh, granulated. (Kidd Creek Mines Ltd.)
- KCCR - Timmins Ont., copper smelter slag (KCCG) + 20% CaO remelted in laboratory, vitrified by quenching. (Kidd Creek Mines Ltd.)
- NMCA - Murdochville, Qué., copper reverberatory slag, fresh, ambient cooled. (Mines Noranda Limitée)
- NRCA - Noranda, Qué., copper reverberatory slag, fresh, ambient cooled. (Noranda Mines Limited)
- NRCR - Noranda, Qué., copper reverberatory slag (NRCA) remelted in laboratory, vitrified by quenching. (Noranda Mines Limited)
- NRCG - Noranda, Qué., copper reverberatory slag, 8 years old, granulated. (Noranda Mines Limited)
- MICG - Mount Isa, Australia, copper reverberatory slag, fresh, granulated. (Mount Isa Mines Limited)
- SWLG - South West Africa/Namibia, copper reverberatory slag, fresh, granulated. (Tsumeb Corporation Limited [Newmont Mining Corporation])
- SWCG - South West Africa/Namibia, lead blast furnace slag, fresh, granulated. (Tsumeb Corporation Limited [Newmont Mining Corporation])
- DOPS - Hamilton, Ont., iron blast furnace slag, fresh, pelletized. (National Slag Limited)
- CECL - Toronto, Ont., Type 10 portland cement clinker, fresh, unground. (St. Lawrence Cement)
- CECLG- Toronto, Ont., Type 10 portland cement clinker + 3% gypsum,fresh, ground. (St. Lawrence Cement)
- BDLO - Belledune, N.B., lead blast furnace slag, old, granulated. (Brunswick Mining and Smelting Corporation)

TABLE 2 Cont.

GLOSSARY FOR SAMPLES

TAILINGS

- KCT - Timmins, Ont., copper smelter tailings. (Not included in study.) (Kidd Creek Mines Ltd.)
- NRT - Noranda, Qué., slag flotation tailings. (Not included in study.) (Noranda Mines Limited)

Table 1), by the Scientific Authority for the study (Dr. E. Douglas), should be noted as it covers the field trips to non-ferrous smelters in: Noranda Quebec, Timmins, Ontario, Sudbury, Ontario; and Belledune, New Brunswick. Additional information received for non-ferrous slags subsequent to the field trips is given in Appendix B. Non-ferrous slags were obtained from five Canadian smelters (lead-slag - Belledune; nickel slag - Inco Sudbury; copper slags - Timmins, Noranda, Murdochville (not visited)) and two foreign smelters (copper slag - Mount Isa; lead and copper slags - Namibia (copper slag not evaluated)). Pelletized iron blast furnace slag, portland cement clinker and gypsum (DOPS, CECL, CECLG) were also obtained for grindability comparisons with non-ferrous slags.

The non-ferrous slags evaluated as sampled were BDLG; ICNA; KCCG; NMCA; NRCA; NRCG; MICG and SWLG. Three non-ferrous slags were subjected to remelting at 1300°C (using a clay graphite crucible) in the laboratory (Ontario Research Foundation) with vitrification (plus lime addition for KCCG) by quenching in excess water (ten water to one slag by weight) before evaluation: ICNA → ICNR; KCCG + 20% CaO → KCCR; and NRCA → NRCR. This resulted in a total of eleven non-ferrous slags for evaluation in the study. Slag samples not consumed during Phase I, or shipped to the CANMET Construction Materials Section, have been stored at Trow Ltd. for use during Phase II.

GRINDING OF NON-FERROUS SLAGS

Grinding of the non-ferrous slags, and comparison cement clinker (with 3% gypsum) and pelletized iron blast furnace slag, was completed using three laboratory ball mills with very similar characteristics (Trow Ltd. staff at Standard Slag Cement). Each ball mill had a capacity of about 35 kg of slag or clinker, and a ball charge of 100 kg (13 mm to 38 mm diameter, production mill steel balls), with care taken with each grinding to keep the charge as consistent as possible in both gradation and mass. The first step with the slags and clinker (with 3% gypsum) was the establishing of grindability curves as shown on Drawings C1 through C15 in Appendix C. This was done by stopping the ball mill at regular intervals during a preliminary grinding trial of slag or clinker, determining the Blaine of a small, representative sample of ground material (i.e. Blaine development with grinding time) and continuing grinding until the desired maximum Blaine was achieved (typically 5000, noting the lengthy grinding time required for BDLG, KCCG and SWLG). The clinker (with 3% gypsum) curve (CECLG) and pelletized slag curve (DOPS) are shown on each non-ferrous slag grindability curve for comparison purposes. Using these grindability curves, separate nominal 2000, 3000, 4000 and 5000 Blaine grinding runs were then made, two for each Blaine.

For "calibration" purposes of grinding time and energy consumption, production level grinding energy requirements during portland cement production (clinker plus 3% gypsum) are about 31 kWh/t to a 3000 Blaine and 43 kWh/t to a 4000 Blaine (35 kWh/t and 50 kWh/t for clinker alone, based on typical F.L. Smidth data). From this grinding energy consumption information, and experience with the laboratory ball mills, a laboratory mill to typical production mill scale-up of 1 minute = 0.4 kWh/t is involved, noting that this depends on the specific laboratory and production mills used and must be confirmed by a potential ground non-ferrous slag producer.

While a full discussion of all test data is reserved for the Analysis and Recommendations section, it would appear that non-ferrous slag grindability is a function of factors such as initial gradation, degree of vitrification and chemistry, with higher glass contents generally requiring significantly more grinding time. Regardless, as the grindability curves in Appendix C indicate, with the exception of ICNA and ICNR, the non-ferrous slags required more grinding energy than DOPS and CECLG, and this was significant in most cases.

Grinding data in terms of initial gradation, actual Blaine finenesses, times to achieve and energy consumption (nominal Blaines of 2000, 3000, 4000 and 5000 adopted for study), specific gravity and percent passing 45 μm are given in Table 3. It should be noted that 5000 Blaine KCCG and SWLG were not prepared given the lengthy time/high energy consumption required for these two non-ferrous slags to reach this Blaine from the grindability curves, Drawings C6 and C13, respectively; i.e. would not be economic in terms of production grinding energy requirements to use 5000 Blaine KCCG, SWLG

TABLE 3

GRINDING DATA FOR SAMPLES

(See Figures In Appendix C for Curves Establishing Grindability)

A. INITIAL GRADATION, % Passing

PASSING	BDLG		ICNA		ICNR		KCCG		KCCR		NMCA Required Initial Crushing from 100 mm	
25.0 mm					100							
					100	*100*						
19.0					96				100			
					96	*96*			100	*100*		
9.5			100		80		100		94		100	
			100	*100*	83	*82*	100	*100*	94	*94*	100	*100*
4.75	100		100		64		99		79		96	
	100	*100*	99	*100*	63	*64*	99	*99*	82	*81*	97	*97*
2.36	99		85		44		93		51		43	
	99	*99*	88	*87*	41	*43*	93	*93*	50	*51*	46	*45*
2.00	99		81		40		88		48		38	
	99	*99*	82	*82*	38	*39*	90	*89*	48	*48*	40	*39*
1.18	95		58		24		57		29		23	
	94	*95*	58	*58*	23	*24*	64	*61*	30	*30*	25	*24*
600 µm	66		34		9		12		10		12	
	59	*63*	32	*33*	10	*10*	18	*15*	9	*10*	14	*13*
300	21		19		4		2		4		7	
	20	*21*	20	*20*	4	*4*	4	*3*	3	*4*	8	*8*
150	6		12		1		.3		2		4	
	5	*6*	12	*12*	2	*2*	1	*.7*	2	*2*	4	*4*
75	2		7		1		.1		.9		2	
	2	*2*	8	*8*	1	*1*	.2	*.2*	.9	*.9*	2	*2*

NOTE: Average given in italics.

TABLE 3 Cont.

GRINDING DATA FOR SAMPLES

(See Figures in Appendix C for Curves Establishing Grindability)

A. INITIAL GRADATION, % Passing, Cont.

PASSING	NRCA		NRCR		NRCG		MICG		SWLG		SWCG	
	Required Initial Crushing from 40 mm						Two samples Blended for Grinding					
25.0 mm												
19.0			100		100		100					
			99	*100*	100	*100*	100	*100*				
9.5	100		97		97		98		100		100	
	100	*100*	95	*96*	100	*99*	99	*99*	100	*100*	100	*100*
4.75	93		84		90		94		99		88	
	90	*92*	81	*83*	87	*89*	99	*97*	98	*99*	89	*89*
2.36	50		50		51		79		93		72	
	46	*48*	50	*50*	46	*49*	90	*85*	89	*91*	70	*71*
2.00	46		45		46		75		92		68	
	41	*44*	43	*44*	41	*44*	88	*82*	87	*90*	66	*67*
1.18	32		20		23		52		71		47	
	25	*29*	20	*20*	21	*22*	66	*59*	66	*69*	47	*47*
600 µm	20		5		8		25		26		21	
	13	*17*	5	*5*	7	*8*	24	*25*	24	*25*	23	*22*
300	12		2		3		16		6		10	
	7	*10*	2	*2*	4	*4*	9	*13*	4	*5*	10	*10*
150	7		1		2		10		1		4	
	4	*6*	.7	*.9*	2	*2*	4	*7*	1	*1*	5	*5*
75	4		.6		1		5		.2		2	
	2	*3*	.4	*.5*	1	*1*	2	*4*	.3	*.3*	2	*2*

TABLE 3 Cont.

GRINDING DATA FOR SAMPLES

(See Figures In Appendix C for Curves Establishing Grindability)

B. GRINDING DATA

BLAINE	BDLG	ICNA	ICNR	KCCG	KCCR	NMCA
	Blaine and Time in Minutes					
2000	2091 75 1966 75 (30) *2029 75*	2085 50 2075 50 (20) *2080 50*	2025 60 (24) *2025 60*	2035 95 1989 95 (38) *2012 95*	1995 55 (22) *1995 55*	2018 50 2049 50 (20) *2034 50*
3000	2936 180 3004 180 (72) *2970 180*	3000 75 3088 75 (30) *3044 75*	3098 75 (30) *3098 75*	2938 195 3014 195 (78) *2976 195*	3168 140 (56) *3168 140*	3098 120 3110 120 (48) *3104 120*
4000	3958 650 4003 655 (261) *3981 653*	4095 120 3998 110 (46) *4047 115*	3935 120 (48) *3935 120*	4032 600 3995 600 (240) *4014 600*	4095 210 (84) *4095 210*	4009 190 3901 190 (76) *3955 190*
5000	5008 1200 5015 1470 (534) *5012 1335*	5033 180 5018 175 (71) *5026 178*	5019 210 (84) *5019 210*	NG	4923 365 (146) *4923 365*	4954 285 4978 285 (114) *4966 285*
SG	4.00	3.54	3.45	3.90	3.68	3.73
% PASSING 45 µm						
2000	76.52	70.57	69.49	69.04	59.65	56.63
3000	85.55	83.68	86.94	79.48	-	82.80
4000	87.59	91.49	91.32	80.92	91.36	87.50
5000	88.52	94.59	95.16	NG	95.31	92.49

NOTES:

1. Two separate values given for Blaine and time since two mills used for each slag, except where single value shown. Averages given in italics.
2. The computed average energy consumption (kWh/t) for production grinding is given in brackets. This is based on a laboratory mill to typical production mill scale-up of 1 minute=0.4 kWh/t for the mills used, and F.L. Smidth data. (Nielsen, H.C. Alsted, Utilization of Fly Ash and Slag in the Cement Industry, Poster, 7th International Congress on the Chemistry of Cement, Paris, 1980.)
3. Two ground samples blended after Blaine values determined, then % passing 45 µm determined.
4. SG determined for 4000 Blaine, using air-comparison pycometer.
5. Nominal Blaine fineness of slags indicated - 2000, 3000, 4000 and 5000.

TABLE 3 Cont.

GRINDING DATA FOR SAMPLES

(See Figures in Appendix C for Curves Establishing Grindability)

B. GRINDING DATA, Cont.

BLAINE	NRCA	NRCR	NRCG	MICG	SWLG	SWCG
		Blaine and Time in Minutes				
2000	2046 50 1911 50 (20)	1985 63 (25)	1978 65 1995 75 (28)	2032 55 (22)	1975 70 (28)	1961 65 (26)
	1979 50	*1985 63*	*1987 70*	*2032 55*	*1975 70*	*1961 65*
3000	3098 115 3116 120 (47)	3150 122 (49)	3023 100 2974 105 (41)	2981 105 (42)	2975 240 (96)	3031 140 (56)
	3107 118	*3150 122*	*2999 103*	*2981 105*	*2975 240*	*3031 140*
4000	3913 190 3940 190 (76)	4138 240 (96)	4052 175 4045 205 (76)	4118 240 (96)	3968 600 (150)	3985 240 (96)
	3927 190	*4138 240*	*4049 190*	*4118 240*	*3968 600*	*3985 240*
5000	4978 325 5020 325 (130)	4936 360 (144)	5159 250 5015 310 (112)	4983 360 (144)	NG	5038 420 (168)
	4999 325	*4936 360*	*5087 280*	*4983 360*		*5038 420*
SG	3.53	3.39	3.50	3.40	3.58	3.40
% PASSING 45 µm						
2000	59.84	62.34	68.04	49.39	63.12	66.04
3000	77.36	90.41	85.42	87.97	71.77	92.42
4000	82.61	92.23	91.68	93.88	73.82	93.79
5000	88.63	92.61	93.09	94.92	NG	94.47

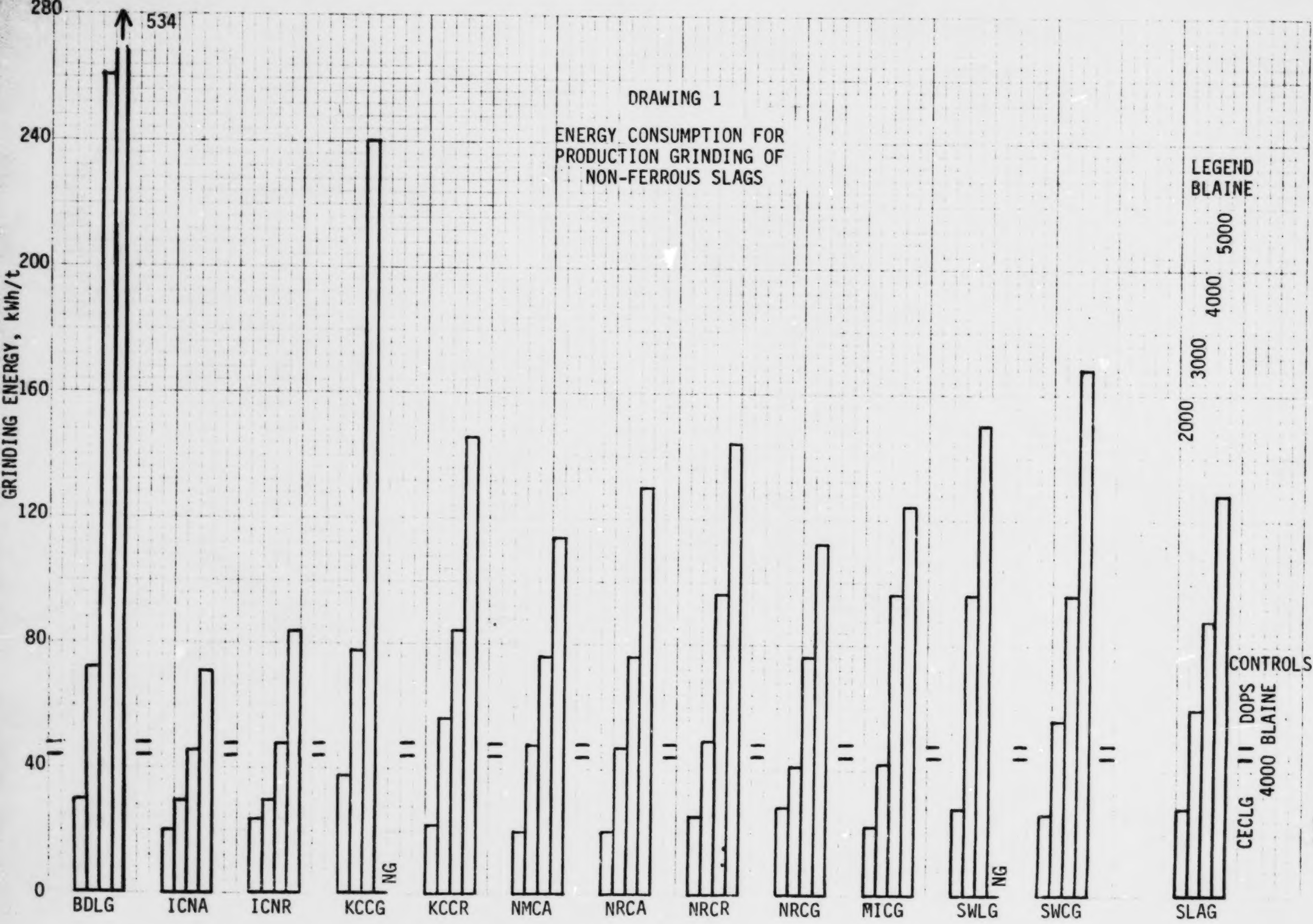
or BDLG (which was prepared to indicate time/energy consumption involved, Drawing C3). For comparison purposes, the estimated energy consumption for production grinding of the eleven non-ferrous slags is given in Drawing 1, along with the 4000 Blaine requirement for CECLG and DOPS.

CHEMICAL AND COMPOSITIONAL CHARACTERISTICS

The chemical and compositional (general) analyses of the non-ferrous slag samples are given in Tables 4 and 5, respectively. Chemical analyses ("whole rock analysis") were completed as follows: a 2000 Blaine sample was fused with lithium carbonate and boric acid; the fused material was then dissolved in hydrochloric and nitric acid, diluted to a known volume, and run on an inductively coupled argon plasma direct reading spectrometer system; and a separate portion of the sample was used for a loss on ignition at 700°C test. As a clay graphite crucible (with some consumption of crucible material) was used when melting ICNA, KCCG and NRCA, this explains the somewhat higher silica and alumina contents of ICNR, KCCR and NRCR.

General XRD analyses are given in Table 5, with a qualitative description of the glass content, as the Siemens X-Ray equipment is still being calibrated for the quantitative analyses of slags. Regardless, it is recommended that the image analysis method (Douglas, E. and Mainwaring, P.R., Fineness, Glass Content and Pozzolanic Activity of Non-ferrous Slags from Canadian Sources, CANMET, July 1983) be adopted for these non-ferrous slags given problems with optical microscopy (opaque particles) and XRD calibration. For instance, optical microscopy glass counts (CSA A363-M1983) indicated 80% to 85% for DOPS, which is typical for this blast furnace slag from previous optical and XRD work; i.e. optical microscopy glass counts are suitable for blast furnace slags. However, for the non-ferrous slags which tend to have opaque particles, optical microscopy glass counts consistently indicated 5% to 10%, which is clearly not the case from Table 5 and the above citation.

It is of interest to note that KCCG is very glassy whereas KCCG remelted + 20% CaO and laboratory quenched, i.e. KCCR, contains more crystalline material. Again, a full discussion of the chemical and compositional characterization data is given in the Analysis and Recommendations section.



DRAWING 1

ENERGY CONSUMPTION FOR PRODUCTION GRINDING OF NON-FERROUS SLAGS

TABLE 4
CHEMICAL ANALYSIS OF SAMPLES
Percent

	BDLG	ICNA	ICNR	KCCG	KCCR	NMCA
SiO_2	21.35	34.91	36.91	26.75	22.10	35.91
	18.46	36.40	37.58	26.33		36.67
	19.91	*35.66*	*37.25*	*26.54*	*22.10*	*36.26*
$A\ell_2O_3$	1.59	5.50	6.07	3.68	2.94	2.14
	1.73	5.68	6.27	3.80		2.17
	1.66	*5.59*	*6.17*	*3.74*	*2.94*	*2.16*
Fe_2O_3	44.87	52.75	52.59	59.68	44.36	48.45
	45.83	53.22	53.55	60.45		49.55
	45.35	*52.99*	*53.07*	*60.07*	*44.36*	*49.00*
CaO	16.59	2.72	2.97	2.10	20.32	7.33
	16.61	2.76	2.97	2.08		7.27
	16.60	*2.74*	*2.97*	*2.09*	*20.32*	*7.30*
MgO	0.69	2.48	2.39	1.60	1.52	1.91
	0.67	2.58	2.47	1.65		1.91
	0.68	*2.53*	*2.43*	*1.63*	*1.52*	*1.91*
Na_2O	0.31	0.91	0.91	0.39	0.23	0.34
	0.20	0.82	0.81	0.27		0.21
	0.26	*0.87*	*0.86*	*0.33*	*0.23*	*0.28*
K_2O	0.14	0.60	0.50	0.16	0.98	1.37
	0.35	0.69	0.73	0.49		1.15
	0.25	*0.65*	*0.62*	*0.33*	*0.98*	*1.26*
TiO_2	0.14	0.32	0.33	0.14	0.12	0.13
	0.14	0.32	0.33	0.14		0.12
	0.14	*0.32*	*0.33*	*0.14*	*0.12*	*0.13*
MnO	0.09	0.07	0.07	0.10	0.08	0.08
	0.09	0.07	0.07	0.10		0.08
	0.09	*0.07*	*0.07*	*0.10*	*0.08*	*0.08*
Cu				1.10		0.48
				1.10		*0.48*
Ni		0.23	0.25			
		0.23	*0.25*			
Pb	4.84					
	4.84					
LOI	-5.54	-5.00	-6.34	-5.55	-4.39	-3.83
	-5.96	-5.64	-6.66	-6.13		-3.99
	-5.75	*-5.32*	*-6.50*	*-5.84*	*-4.39*	*-3.91*
S	1.30	1.46	1.39	1.28	0.45	1.01
	1.30	*1.46*	*1.39*	*1.28*	*0.45*	*1.01*

NOTES:

1. Analyses by Technical Service Laboratories.
2. Major oxides by whole rock analyses - inductively coupled argon plasma direct reading spectrometer system.
3. P_2O_5 not determined due to severe spectral interference from Cu and Zn.
4. Total iron expressed as Fe_2O_3, negative LOI indicates FeO present.
5. LOI at 700°C.
6. S by LECO.
7. Averages given in italics.

TABLE 4 Cont.
CHEMICAL ANALYSIS OF SAMPLES
Percent

	NRCA	NRCR	NRCG	MICG	SWLG	SWCG	DOPS	CECL
SiO_2	33.92	36.66	34.17	42.13	26.93	37.21	38.33	19.61
	35.09	36.89	34.64	43.39	27.37	38.02	40.36	21.24
	34.51	*36.78*	*34.41*	*42.76*	*27.15*	*37.62*	*39.35*	*20.43*
AL_2O_3	6.43	7.06	6.71	3.32	3.89	4.31	5.73	4.45
	6.67	7.26	6.93	3.38	3.87	4.42	5.45	3.20
	6.55	*7.16*	*6.82*	*3.35*	*3.88*	*4.37*	*5.59*	*3.83*
Fe_2O_3	49.14	49.81	48.88	45.29	28.11	45.68	1.37	2.47
	49.86	50.19	50.39	46.28	28.90	46.93	1.22	2.50
	49.50	*50.00*	*49.64*	*45.79*	*28.51*	*46.31*	*1.30*	*2.49*
CaO	2.16	1.93	1.66	8.20	20.35	6.06	36.46	63.81
	2.23	1.93	1.66	8.11	20.28	5.98	37.25	65.01
	2.20	*1.93*	*1.66*	*8.16*	*20.32*	*6.02*	*36.86*	*64.41*
MgO	1.44	1.46	1.87	2.88	4.75	2.73	12.20	2.45
	1.52	1.51	1.94	2.96	4.91	2.83	12.64	2.50
	1.48	*1.49*	*1.91*	*2.92*	*4.83*	*2.78*	*12.42*	*2.48*
Na_2O	1.05	1.11	1.36	0.29	0.62	0.56	0.42	0.36
	0.98	1.00	1.31	0.13	0.50	0.43	0.26	0.12
	1.02	*1.06*	*1.34*	*0.21*	*0.66*	*0.50*	*0.34*	*0.24*
K_2O	1.01	0.95	0.57	0.63	0.56	0.83	0.64	1.14
	1.01	0.96	0.75	0.66	0.50	0.92	0.57	0.97
	1.01	*0.96*	*0.66*	*0.65*	*0.53*	*0.88*	*0.61*	*1.06*
TiO_2	0.34	0.37	0.28	0.24	0.21	0.29	0.30	0.28
	0.35	0.38	0.28	0.24	0.21	0.29	0.31	0.28
	0.35	*0.39*	*0.28*	*0.24*	*0.21*	*0.29*	*0.31*	*0.28*
MnO	0.09	0.09	0.08	0.05	0.30	0.16	1.33	0.04
	0.10	0.09	0.08	0.05	0.31	0.16	1.37	0.04
	0.10	*0.09*	*0.08*	*0.05*	*0.31*	*0.16*	*1.35*	*0.04*
Cu	0.43	0.41	0.48					
	0.43	*0.41*	*0.48*					
Ni								
Pb				0.06	1.92	0.64		
				0.06	*1.92*	*0.64*		
LOI	-4.78	-5.75	-4.63	-3.27	-2.77	-4.26	-0.38	0.96
	-5.70	-6.50	-4.83	-3.61	-2.97	-4.56	0.00	0.38
	-5.24	*-6.13*	*-4.73*	*-3.44*	*-2.87*	*-4.41*	*-0.19*	*0.67*
S	1.20	1.13	1.00	0.56	0.67	0.90	1.15	0.96
	1.20	*1.13*	*1.00*	*0.56*	*0.67*	*0.90*	*1.15*	*0.96*

TABLE 5

XRD ANALYSIS OF SAMPLES

SAMPLE	RESULTS
1. BDLG	Glassy slag. The crystalline material comprises a melilite-type of mineral and wustite (FeO).
2. INCA	Contains abundant crystalline material. The main minerals are fayalite and magnetite.
3. ICNR	Still contains substantial crystalline material. The only material present is fayalite.
4. KCCG	Very glassy slag. The small amount of crystalline material comprises olivine (probably fayalite) and magnetite.
5. KCCR	Contains more crystalline material than the fresh, granulated slag. The crystalline material comprises fayalite.
6. NMCA	Contains abundant crystalline material. The main minerals present are olivine, diopside and magnetite.
7. NRCA	Contains abundant crystalline material. The main minerals present are fayolite (Fe-rich olivine) and magnetite (Fe_3O_4).
8. NRCR	Very glassy slag. The small amount of crystalline material is fayalite.
9. NRCG	Contains abundant crystalline material. The main minerals present are fayalite and magnetite.
10. MICG	Contains abundant crystalline material. The main mineral present is quartz; also contains a few percent fayolite.
11. DOPS	Contains moderate amount of crystalline material. The main constituents are akermanite and hematile (Fe_2O_3).

NOTES: 1. Analyses by Building Research Establishment.
2. Siemens X-Ray equipment used for general analysis for main minerals. Machine being calibrated for quantitative analysis of slags.

HYRAULIC AND POZZOLANIC INDEX TESTING
(ASTM C595 AND C989)

The separtely ground slag index, blended cement hydraulic index, blended cement pozzolanic index and compressive strength development of mortar cubes with curing time for nominal 2000, 3000, 4000 and 5000 Blaine finenesses of the eleven non-ferrous slags at various blends and curing conditions were completed in accordance with ASTM C109, C595 and C989 (recently developed by ASTM Subcommittee E38.06 for separately ground slag cement). In addition, activator influences using NaOH, and cement kiln dust replacement of Portland cement, were investigated. This testing resulted in considerable data which are summarized in Tables 6-8 with technical notes and Drawings 2-14- with detailed discussion given in the next section.

Table 6 and Drawings 2-13 — Compressive strength development (ASTM C989) for 1, 7, 28 and 90 days curing at 23°C, with ground non-ferrous slag (2000, 3000, 4000, 5000 nominal Blaine) at a 50 percent absolute volume replacement of Type 10 Portland cement (noting that the controls are based on a C989 type cement, a somewhat conservative approach). This data appear to be the most significant generated for the pozzolanic and hydraulic index testing, with the positive influence of higher Blaines for ambient curing conditions showing up clearly. There is a wide range in the strength index (Drawing 13) with no ground non-ferrous slag approaching unity (i.e. "portland cement equivalency"). However, several appear to have significant Portland cement replacement potential at the 4000 Blaine level (5000 Blaine typically better, but much greater grinding energy required for relative improvement in strength): BDLG (excellent 90 day strengths); KCCG; KCCR (noting 20% CaO added and economics of this questionable); NRCR; MICG; and SWLG.

Table 7 and Drawing 14 — Compressive strength development (ASTM C595) for 28 days curing at 38°C (i.e. accelerated curing) with ground non-ferrous slag (2000, 3000, 4000, 5000 nominal Blaine) at a 20, 35, 50 and 70 percent absolute volume replacement of Type 10 Portland cement. The poz-

TABLE 6
"C109" MORTAR CUBE COMPRESSIVE STRENGTH DEVELOPMENT AT 23°C
50 PERCENT ABSOLUTE VOLUME REPLACEMENT OF PORTLAND CEMENT
(ASTM C989)

		BDLG	ICNA	ICNR	KCCG	KCCR	NMCA
CONTROLS	7D	31.3 111	31.3 111	31.3 111	31.3 111	31.3 111	31.3 111
	28D	36.8	36.8	36.8	36.8	36.8	36.8
2000							
Days 1	MPa	7.0 105	3.9 105	4.8 105	4.6 111	4.7 105	3.1 109
7		10.8	8.7	10.7	9.0	9.1	8.8
28		19.1	14.2	17.8	19.0	17.0	12.0
90		29.5	21.6	23.0	25.1	21.7	16.7
3000							
Days 1	MPa	8.5 115	4.0 109	5.0 108	4.9 112	5.0 110	4.7 115
7		12.4	8.8	10.8	10.3	11.0	10.9
28		23.5	15.6	19.3	25.7	21.8	15.7
90		37.9	26.0	26.1	29.7	26.5	20.6
4000							
Days 1	MPa	12.1 113	4.1 113	5.2 106	5.6 107	6.3 115	5.0 113
7		14.1	10.0	11.7	10.5	12.5	12.0
28		29.8	17.5	21.7	26.7	24.8	19.2
90		40.2	28.3	26.6	31.4	31.1	23.2
5000							
Days 1	MPa	14.1 113	4.4 107	7.0 106	NG	7.5 113	5.1 112
7		18.2	11.2	13.1		16.0	12.9
28		28.7	25.0	26.2		30.4	21.3
90		43.7	33.1	31.7		38.9	26.8
INDEX 7 Day/28 Day							
2000		0.35/0.52	0.28/0.39	0.34/0.48	0.29/0.52	0.29/0.46	0.28/0.33
3000		0.40/0.64	0.28/0.42	0.35/0.52	0.33/0.70	0.35/0.59	0.35/0.43
4000		0.45/0.81	0.32/0.48	0.37/0.59	0.34/0.73	0.40/0.67	0.38/0.52
5000		0.58/0.78	0.36/0.68	0.42/0.71	NG	0.51/0.83	0.41/0.58

NOTES:

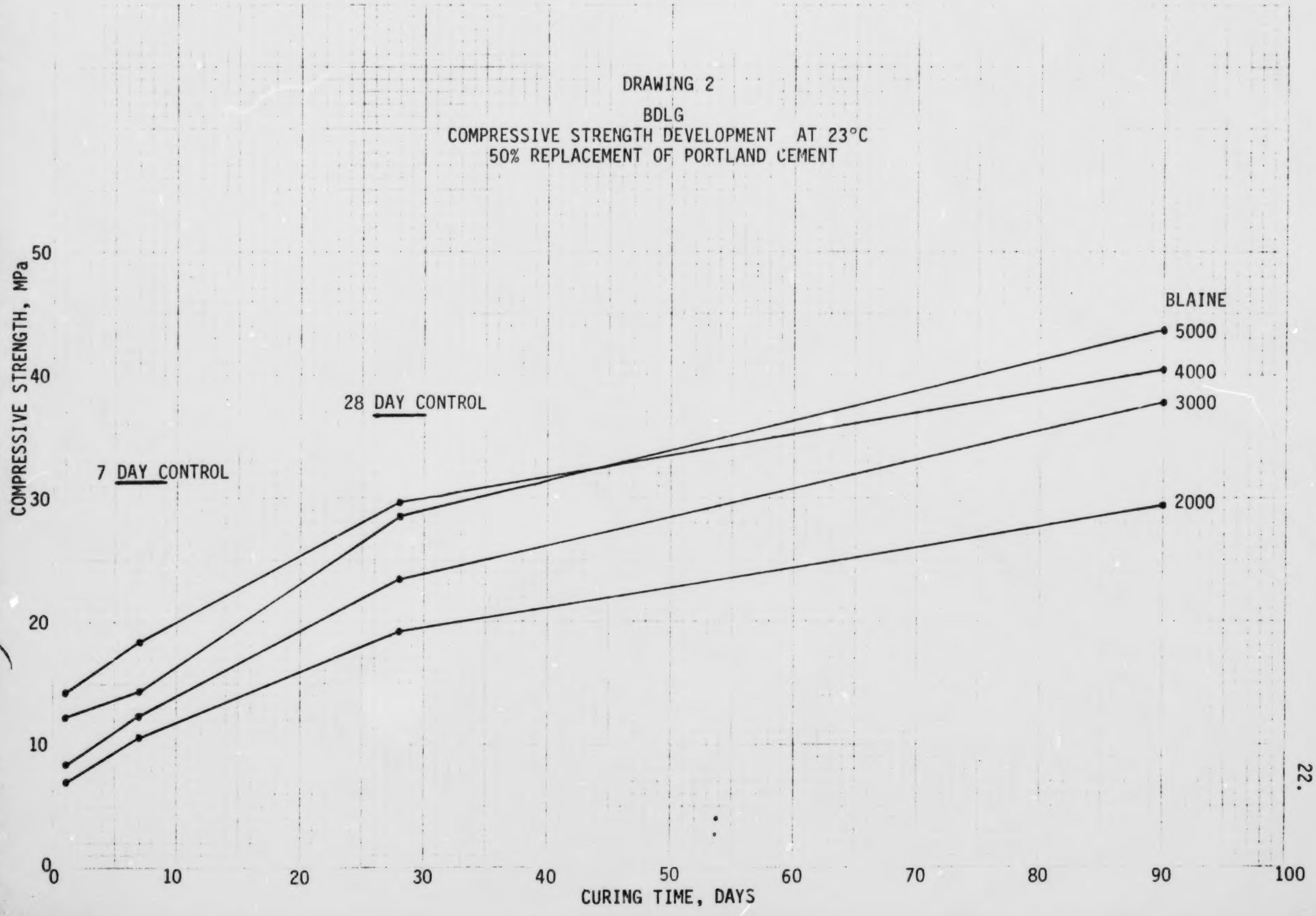
1. Portland cement meets requirements of ASTM "Specifications for Ground Blast-Furnace Slag for Use in Concrete and Mortars", as developed by E38.06.02. (Peerless cement intermediate between Type 1 and Type 2.)
2. Nominal Blaine fineness of slags indicated-2000, 3000, 4000 and 5000.
3. All compressive strengths in MPa, with flow indicated for each mix.
4. NG indicates slag was not ground to this fineness.
5. INDEX is the 7 day and 28 day ratio of slag/cement mortar cube compressive strength to cement mortar cube (control) compressive strength.

TABLE 6 Cont.

"C109" MORTAR CUBE COMPRESSIVE STRENGTH DEVELOPMENT AT 23°C
50 PERCENT ABSOLUTE VOLUME REPLACEMENT OF PORTLAND CEMENT

(ASTM C989)

	NRCA	NRCR	NRCG	MICG	SWLG
CONTROLS 7	31.3 111	31.3 111	31.3 111	31.3 111	31.3 111
28	36.8	36.8	36.8	36.8	36.8
2000					
Days 1 MPa	3.3 115	4.3 110	3.8 111	3.3 105	3.3 107
7	9.2	9.3	7.4	7.8	11.8
28	14.4	16.2	13.8	12.6	20.1
90	19.4	21.1	18.5	17.8	29.7
3000					
Days 1 MPa	3.3 113	5.0 106	4.0 115	4.0 109	3.4 115
7	9.5	12.6	8.6	8.8	14.4
28	18.0	24.6	14.2	19.9	25.9
90	25.9	32.0	24.3	24.9	39.6
4000					
Days 1 MPa	4.3 107	5.6 107	4.7 112	4.2 105	7.9 109
7	11.0	14.8	10.3	12.2	16.9
28	20.5	30.5	21.6	26.7	33.4
90	29.2	37.3	29.1	34.8	43.2
5000					
Days 1 MPa	4.5 105	5.9 115	5.3 106	4.3 109	NG
7	11.2	17.1	11.5	13.2	
28	24.2	35.2	25.9	29.0	
90	32.0	38.3	32.8	36.0	
INDEX 7 Day/28 Day					
2000	0.30/0.39	0.30/0.44	0.24/0.38	0.25/0.34	0.38/0.55
3000	0.30/0.49	0.40/0.67	0.27/0.39	0.28/0.54	0.46/0.70
4000	0.35/0.56	0.47/0.83	0.33/0.59	0.39/0.73	0.54/0.91
5000	0.36/0.66	0.55/0.96	0.37/0.70	0.42/0.79	NG

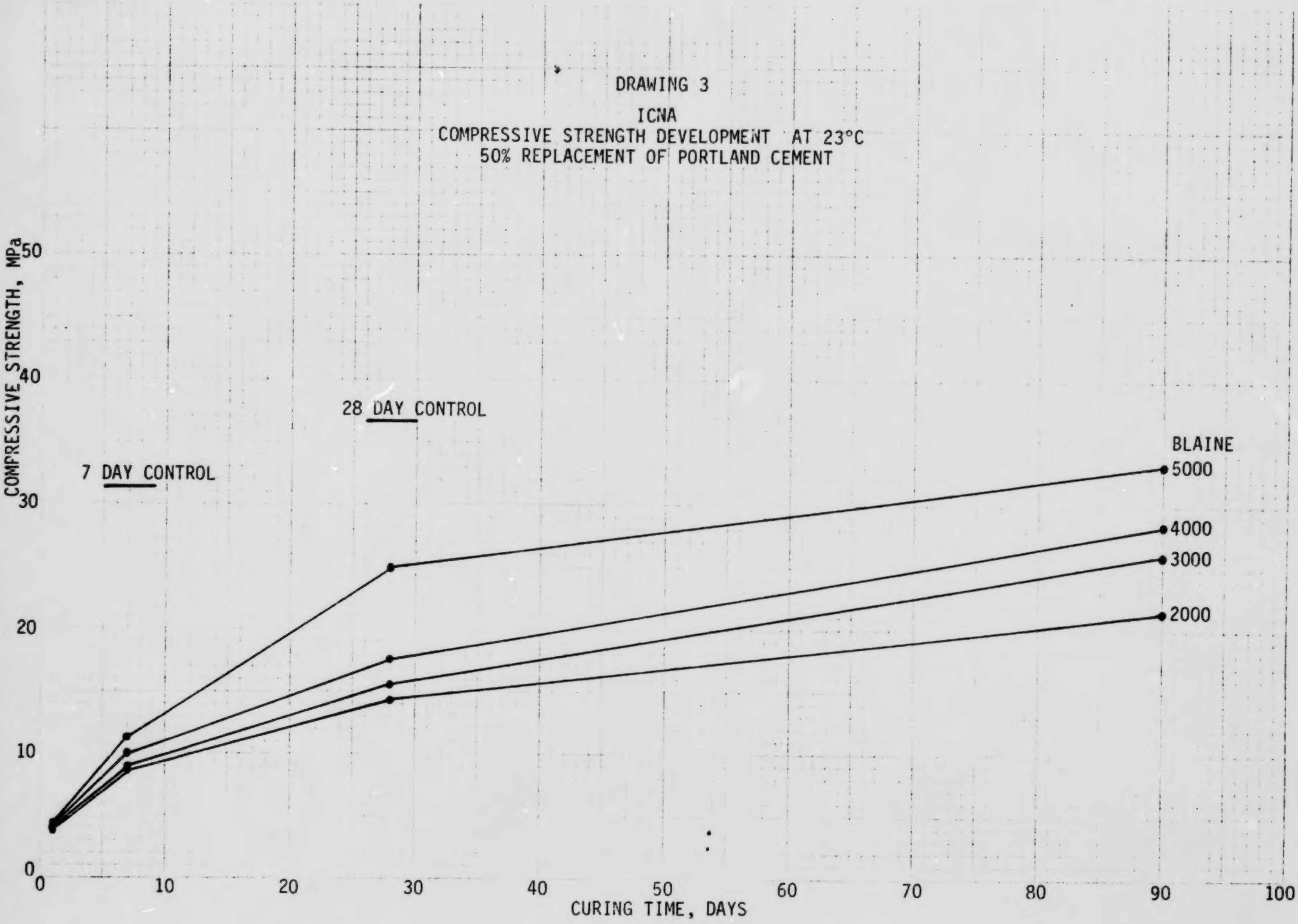


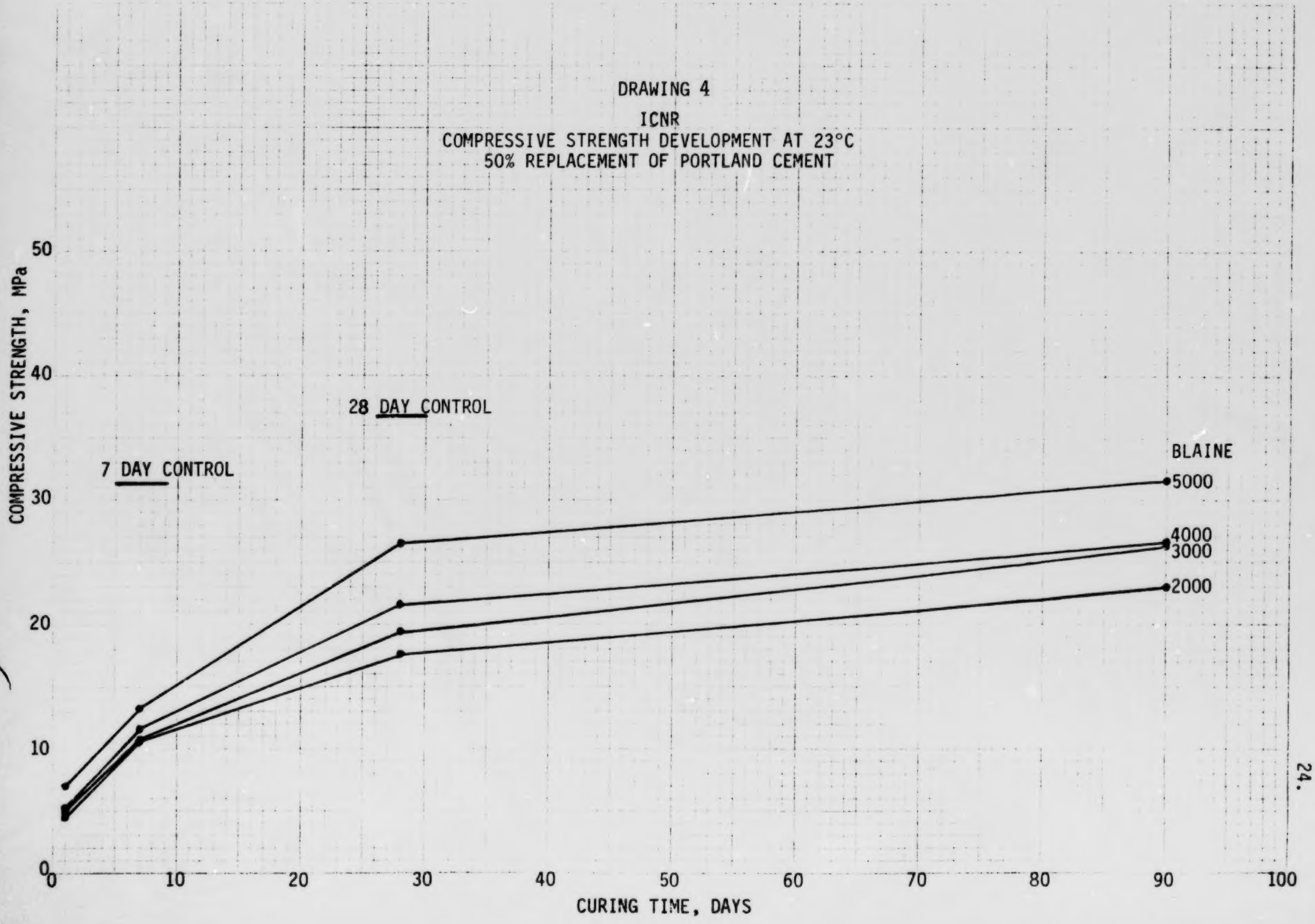
DRAWING 2

BDLG
COMPRESSIVE STRENGTH DEVELOPMENT AT 23°C
50% REPLACEMENT OF PORTLAND CEMENT

DRAWING 3

ICNA
COMPRESSIVE STRENGTH DEVELOPMENT AT 23°C
50% REPLACEMENT OF PORTLAND CEMENT



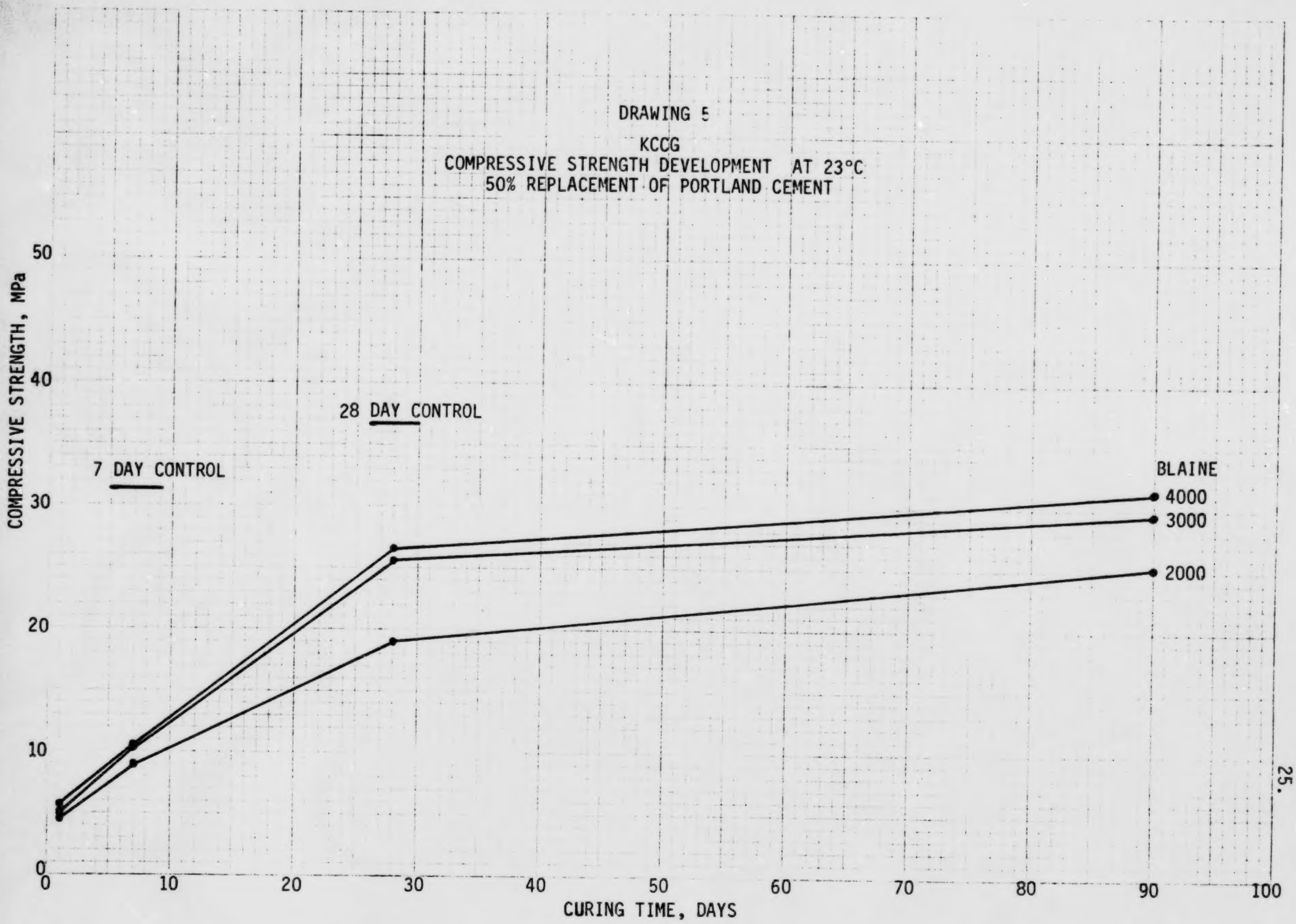


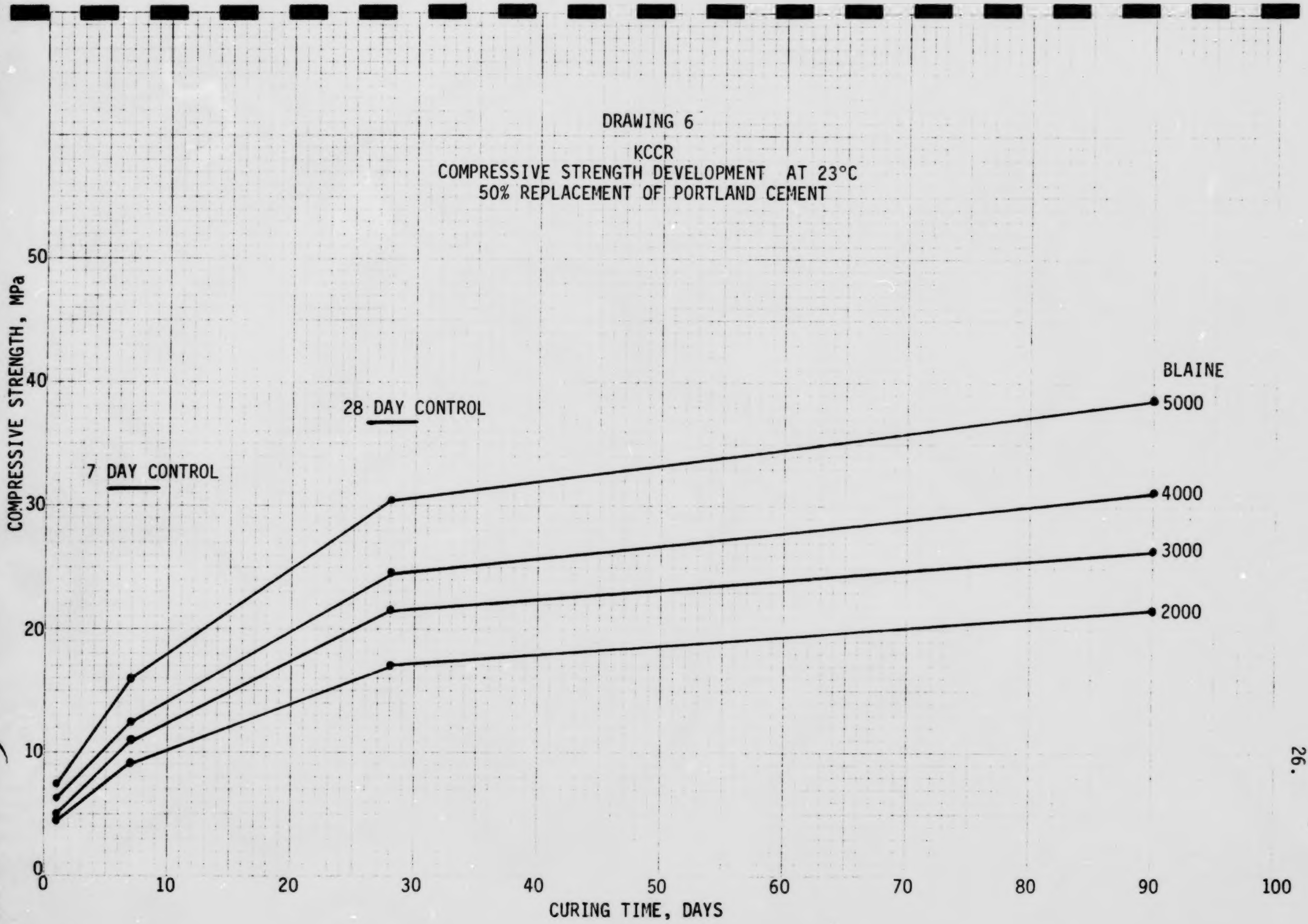
DRAWING 4

ICNR
COMPRESSIVE STRENGTH DEVELOPMENT AT 23°C
50% REPLACEMENT OF PORTLAND CEMENT

DRAWING 5

KCCG
COMPRESSIVE STRENGTH DEVELOPMENT AT 23°C
50% REPLACEMENT OF PORTLAND CEMENT



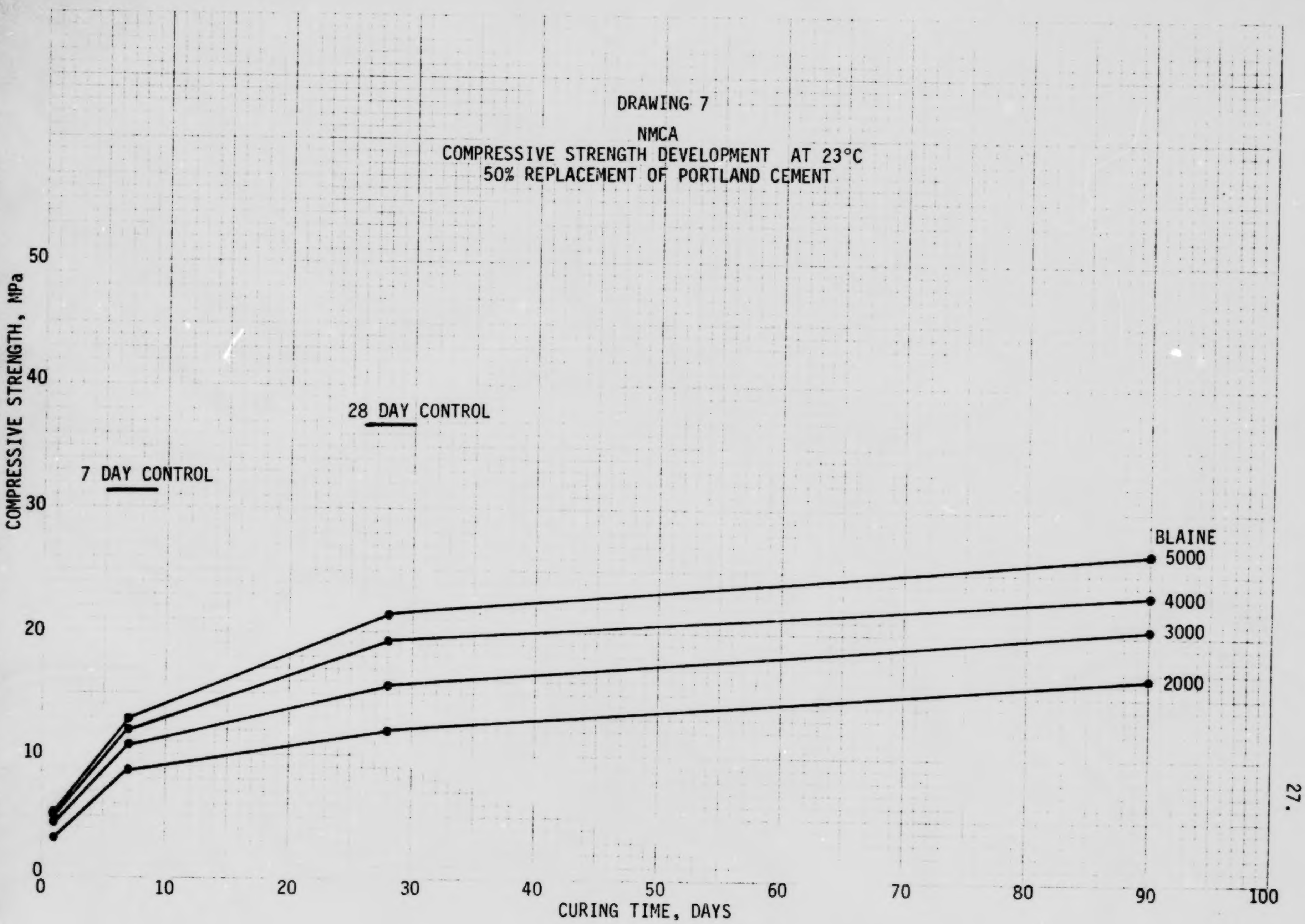


DRAWING 6

KCCR
COMPRESSIVE STRENGTH DEVELOPMENT AT 23°C
50% REPLACEMENT OF PORTLAND CEMENT

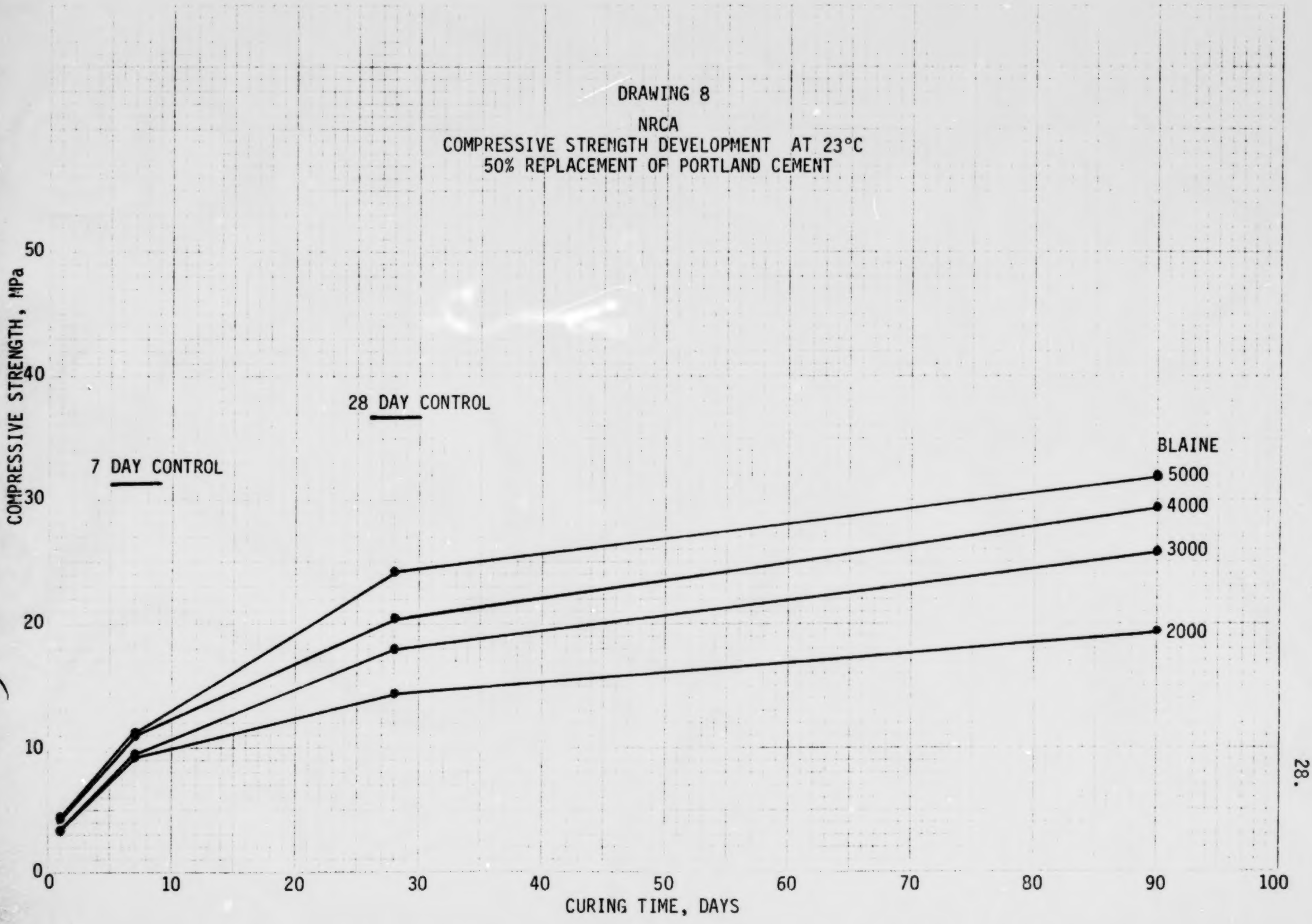
DRAWING 7

NMCA
COMPRESSIVE STRENGTH DEVELOPMENT AT 23°C
50% REPLACEMENT OF PORTLAND CEMENT



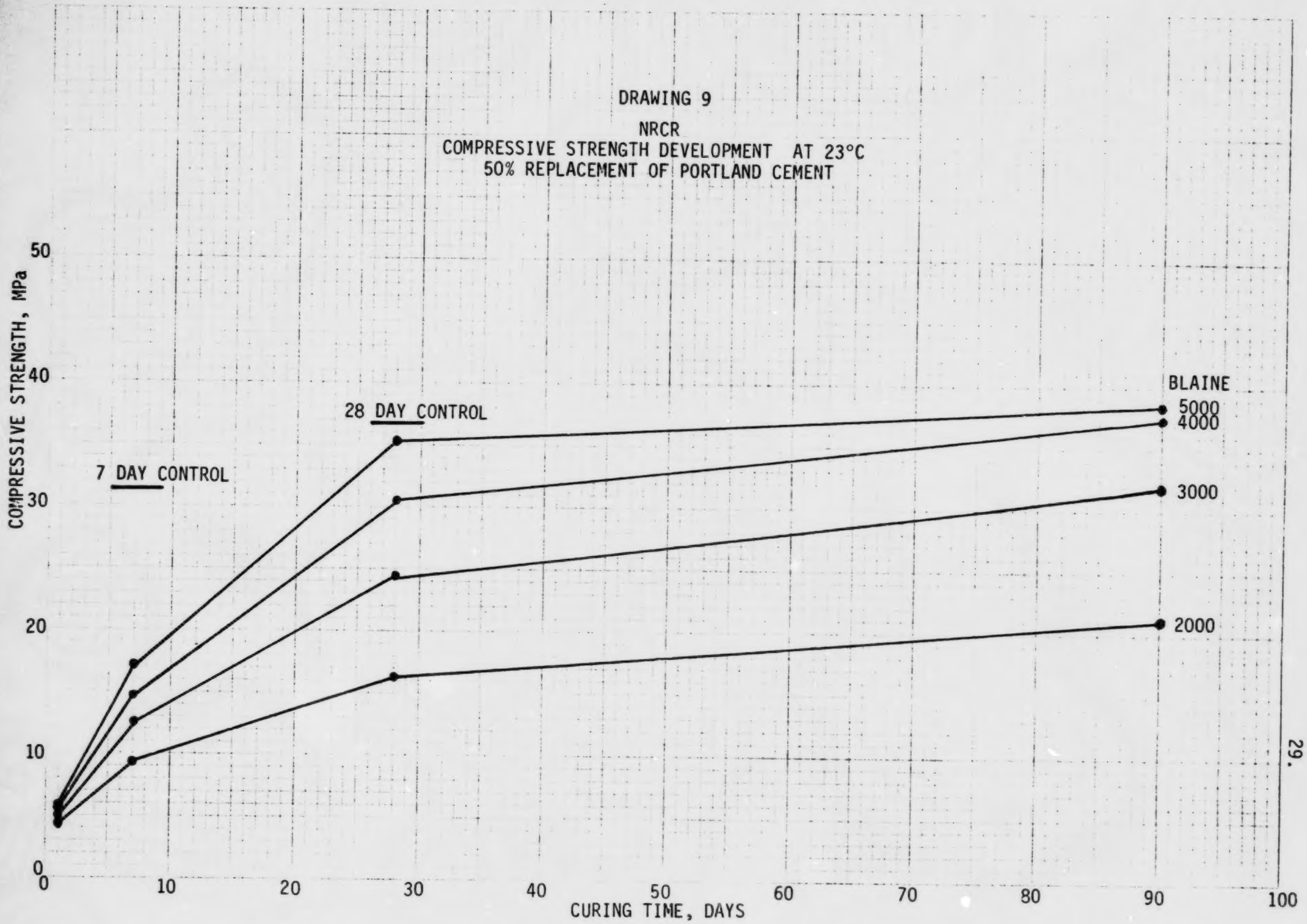
DRAWING 8

NRCA
COMPRESSIVE STRENGTH DEVELOPMENT AT 23°C
50% REPLACEMENT OF PORTLAND CEMENT



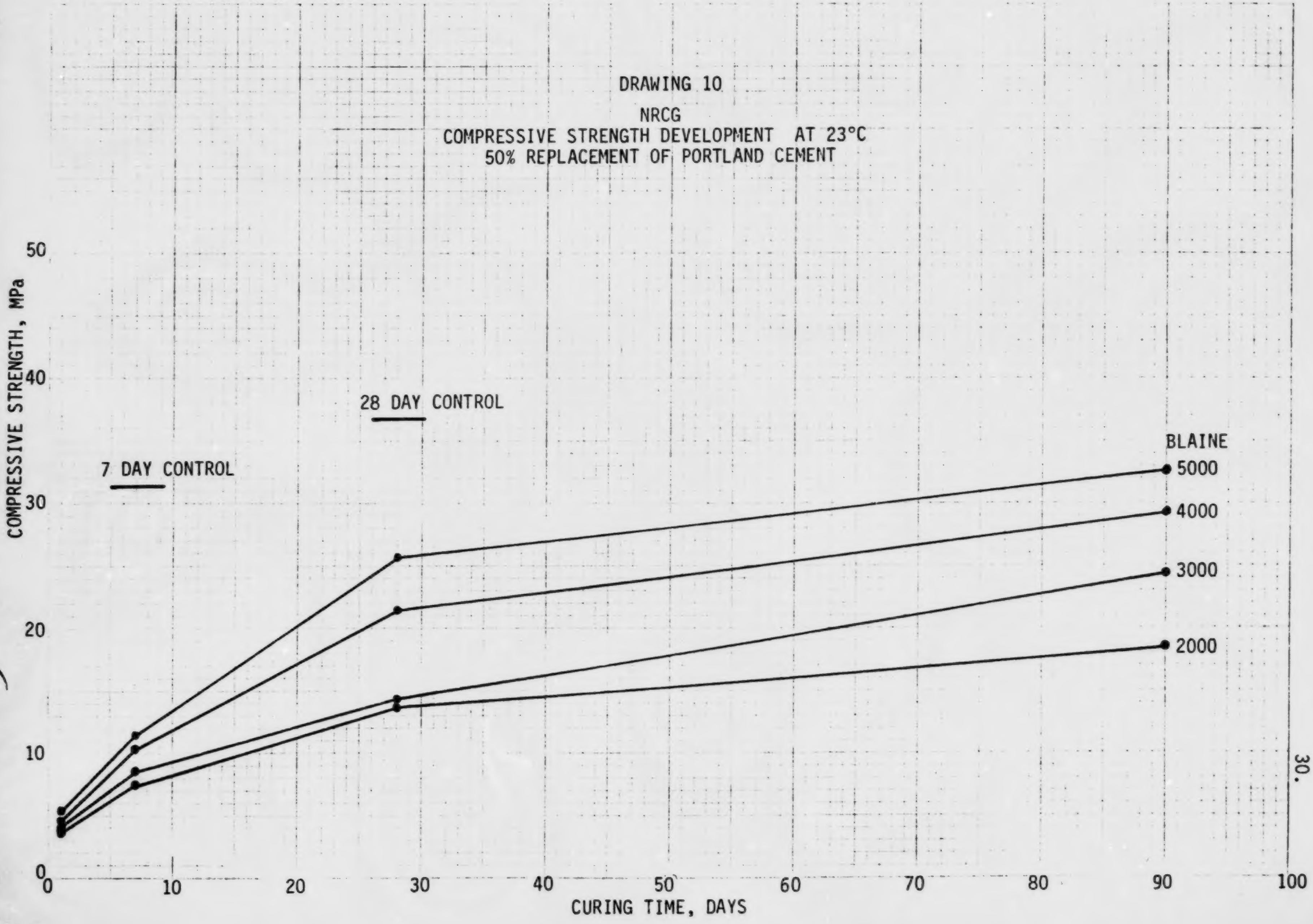
DRAWING 9

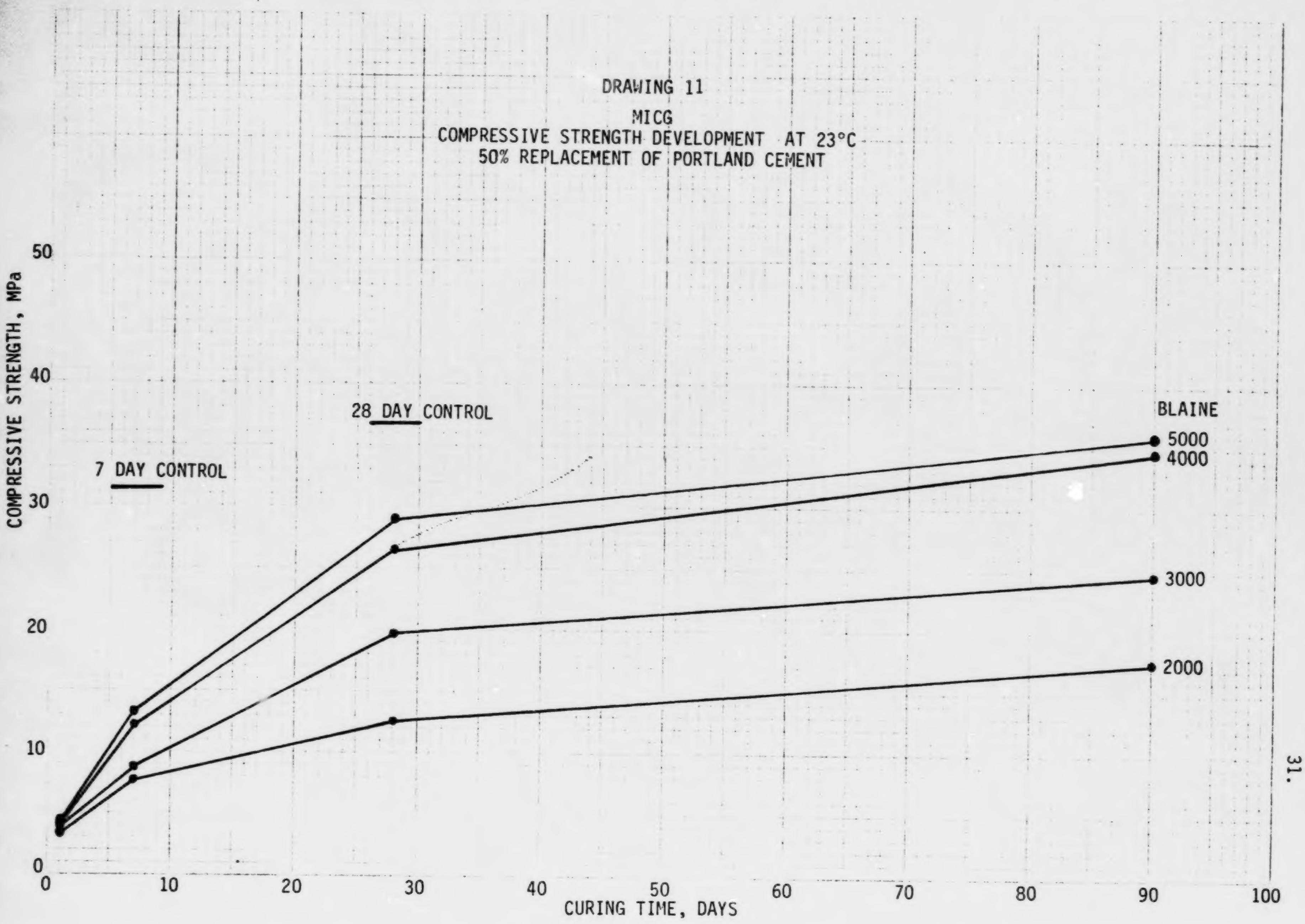
NRCR
COMPRESSIVE STRENGTH DEVELOPMENT AT 23°C
50% REPLACEMENT OF PORTLAND CEMENT



DRAWING 10

NRCG
COMPRESSIVE STRENGTH DEVELOPMENT AT 23°C
50% REPLACEMENT OF PORTLAND CEMENT



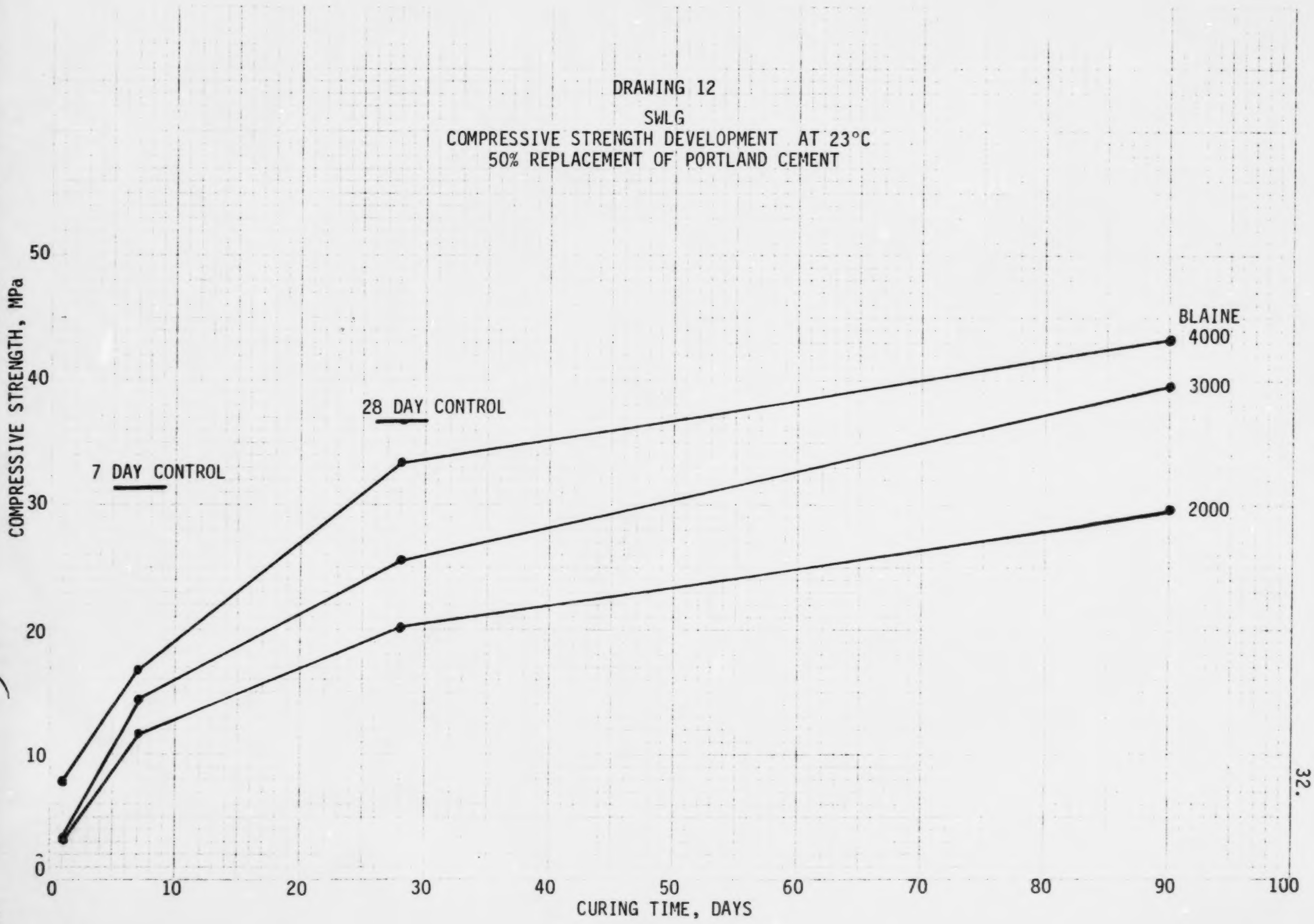


DRAWING 11

MICG
COMPRESSIVE STRENGTH DEVELOPMENT AT 23°C
50% REPLACEMENT OF PORTLAND CEMENT

DRAWING 12

SWLG
COMPRESSIVE STRENGTH DEVELOPMENT AT 23°C
50% REPLACEMENT OF PORTLAND CEMENT



DRAWING 13

INDEX FOR COMPRESSIVE STRENGTH DEVELOPMENT
7 DAYS AND 28 DAYS CURING AT 23°C
50% REPLACEMENT OF PORTLAND CEMENT

INDEX is the ratio of slag/cement mortar cube compressive strength to cement mortar cube (control) compressive strength.

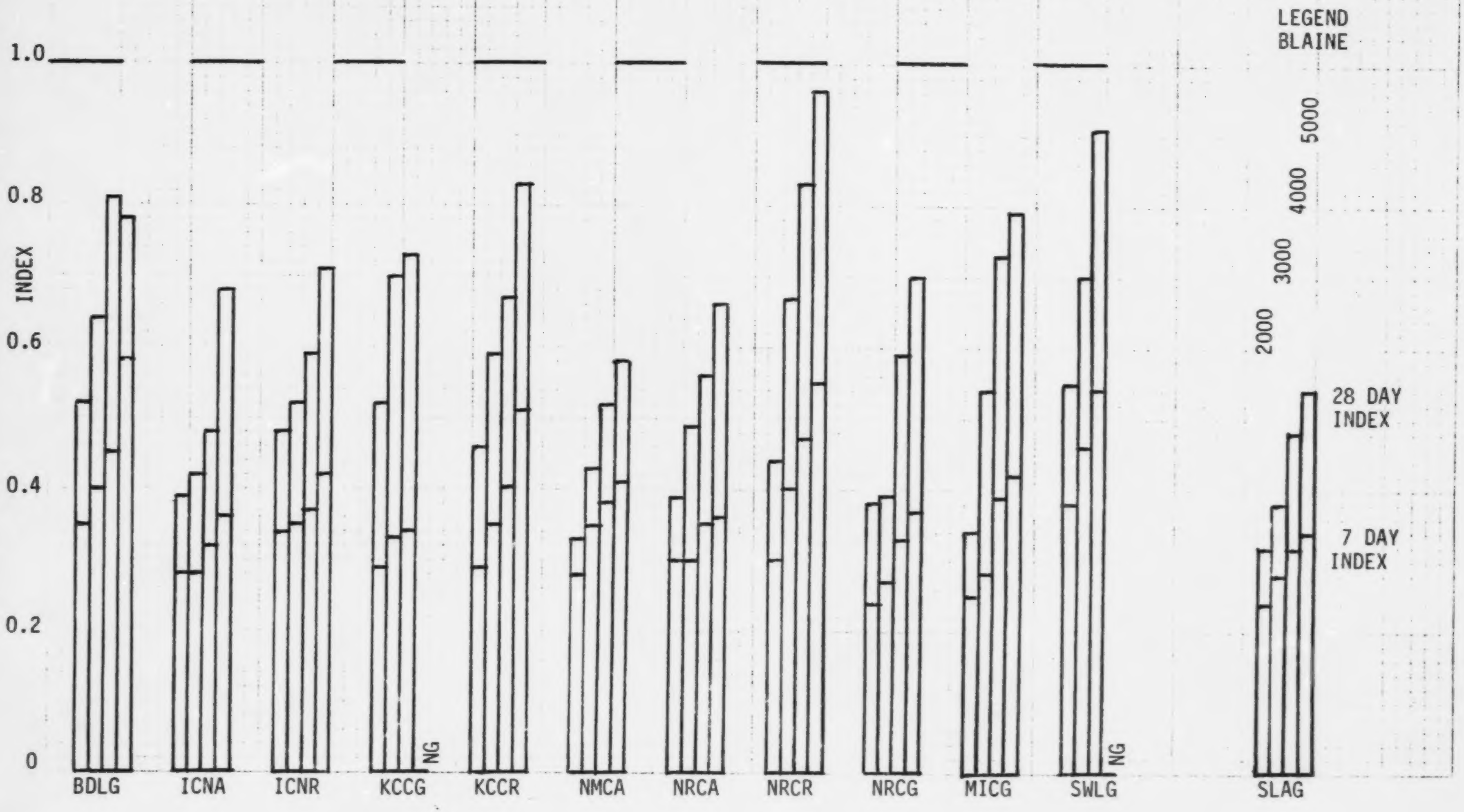


TABLE 7
"C109" MORTAR CUBE COMPRESSIVE STRENGTH DEVELOPMENT, 28 DAYS CURING AT 38°C
20, 35, 50 AND 70 PERCENT ABSOLUTE REPLACEMENT OF PORTLAND CEMENT

		BDLG	ICNA	ICNR	KCCG	KCCR	NMCA
CONTROL		29.6 111	29.6 111	29.6 111	29.6 111	29.6 111	29.6 111
20%							
2000	MPa	21.4 110	22.5 105	17.1 106	16.0 113	20.7 111	20.8 115
3000		22.7 114	24.0 106	20.9 115	23.4 112	23.6 114	22.1 115
4000		30.5 114	24.8 112	23.8 112	25.4 114	25.8 114	24.6 115
5000		32.3 105	30.2 106	26.3 115	NG	27.5 114	25.0 115
35%							
2000	MPa	14.7 112	18.9 106	16.4 105	14.2 111	17.0 106	14.6 105
3000		16.1 106	21.5 112	19.7 110	20.4 114	21.3 113	19.8 112
4000		28.5 107	24.5 106	23.0 114	20.3 112	22.5 107	20.8 111
5000		30.0 106	29.1 111	25.9 109	NG	24.1 109	24.6 105
50%							
2000	MPa	13.7 108	18.3 114	13.9 108	9.5 109	13.8 105	11.3 112
3000		15.0 114	19.1 115	17.3 105	15.5 114	18.0 113	18.0 108
4000		20.6 108	21.4 105	20.9 115	17.8 105	18.8 107	19.7 110
5000		22.3 108	28.7 106	25.4 108	NG	19.6 114	23.3 105
70%							
2000	MPa	7.3 105	11.9 108	7.8 109	7.2 107	9.7 105	7.0 115
3000		7.9 115	12.8 115	12.1 105	8.5 107	12.8 106	10.5 110
4000		8.5 108	17.1 113	13.0 105	9.1 111	13.4 105	13.9 106
5000		10.8 107	19.8 105	16.8 108	NG	15.5 115	15.6 107
PI/HI							
2000		0.50/0.25	0.64/0.40	0.55/0.26	0.48/0.24	0.57/0.33	0.49/0.24
3000		0.54/0.27	0.73/0.43	0.67/0.41	0.69/0.29	0.72/0.43	0.67/0.35
4000		0.96/0.29	0.83/0.58	0.78/0.44	0.69/0.31	0.76/0.45	0.70/0.47
5000		1.01/0.36	0.98/0.67	0.88/0.57	NG	0.81/0.52	0.83/0.53

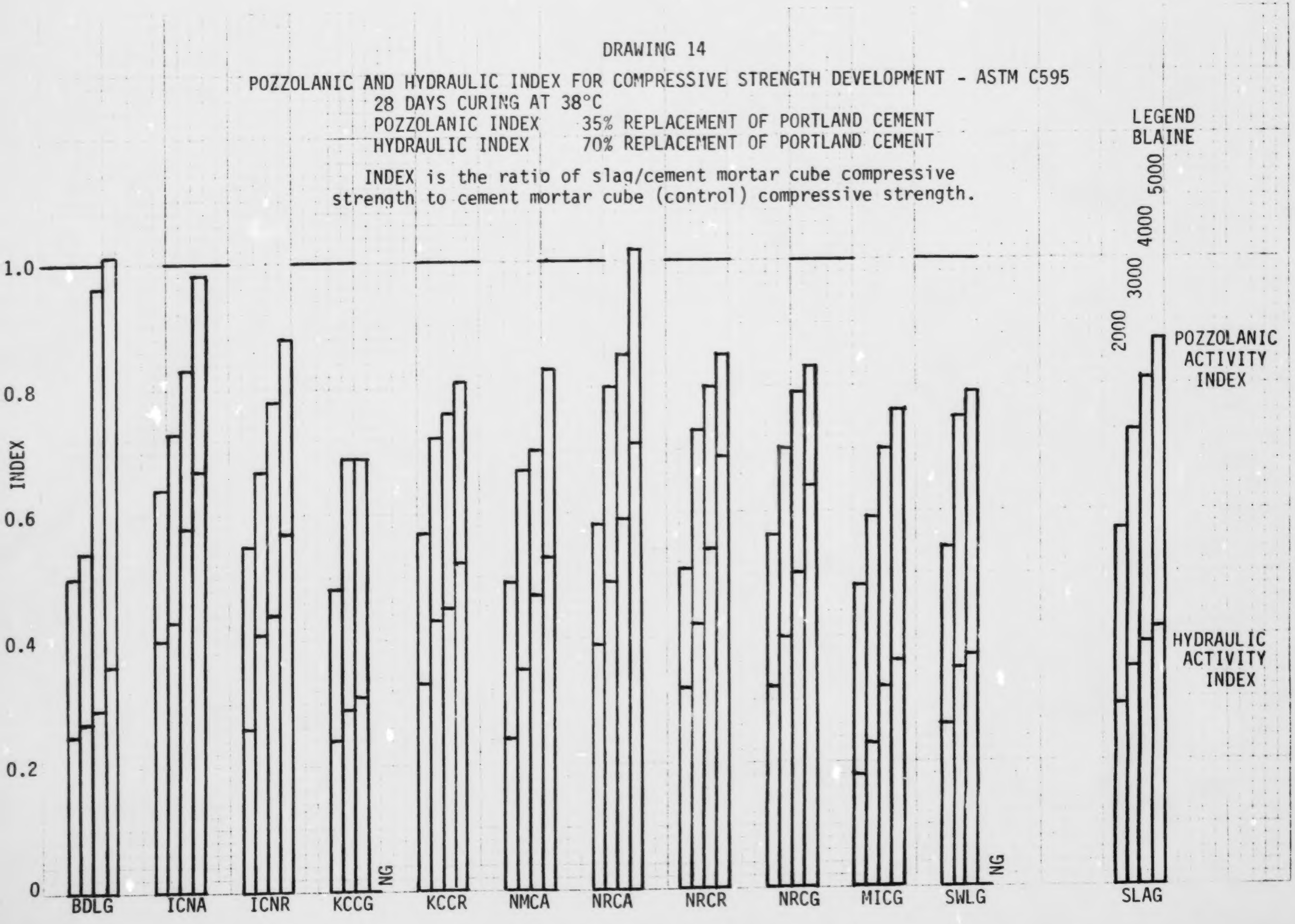
NOTES:
1. Portland cement meets requirements of CSA Canadian Type 10. (Canada Cement Lafarge Type 10.)
2. Nominal Blaine fineness of slags indicated - 2000, 3000, 4000 and 5000.
3. All compressive strengths in MPa, with flow indicated for each mix.
4. NG indicates slag was not ground to this fineness.
5. PI and HI are the pozzolanic (pozzanic activity index) and hydraulic (slag activity index) index with cement, respectively, in accordance with ASTM C595.

TABLE 7 Cont.

"C109" MORTAR CUBE COMPRESSIVE STRENGTH DEVELOPMENT, 28 DAYS CURING AT 38°C

20, 35, 50 AND 70 PERCENT ABSOLUTE REPLACEMENT OF PORTLAND CEMENT

		NRCA	NRCR	NRCG	MICG	SWLG
CONTROL		29.6 111	29.6 111	29.6 111	29.6 111	29.6 111
20%						
2000	MPa	21.7 115	19.3 109	20.2 106	18.3 105	19.1 114
3000		25.2 114	23.9 110	20.6 105	22.5 113	23.1 115
4000		26.5 115	25.7 105	24.6 106	24.5 106	24.4 108
5000		30.8 107	27.6 105	26.2 115	26.8 111	NG
35%						
2000	MPa	17.2 105	15.1 112	16.6 114	14.1 109	15.9 105
3000		23.7 105	21.5 113	20.8 110	17.5 111	22.3 114
4000		25.3 114	23.8 105	23.5 110	20.6 113	23.4 107
5000		30.3 105	25.3 111	24.3 115	22.6 110	NG
50%						
2000	MPa	14.6 110	13.4 106	13.3 111	8.6 105	12.6 106
3000		22.7 105	20.2 114	17.3 110	14.0 109	16.9 106
4000		23.9 105	23.6 108	21.3 106	15.5 109	21.9 105
5000		26.9 115	24.4 113	22.0 105	20.2 112	NG
70%						
2000	MPa	11.5 113	9.7 115	9.6 105	5.2 115	7.8 107
3000		14.5 115	12.5 109	11.7 115	6.8 114	10.5 111
4000		17.5 111	16.0 105	14.7 110	9.4 115	10.9 115
5000		21.0 107	20.4 106	18.8 114	10.7 115	NG
PI/HI						
2000		0.58/0.39	0.51/0.32	0.56/0.32	0.48/0.18	0.54/0.26
3000		0.80/0.49	0.73/0.42	0.70/0.40	0.59/0.23	0.75/0.35
4000		0.85/0.59	0.80/0.54	0.79/0.50	0.70/0.32	0.79/0.37
5000		1.02/0.71	0.85/0.69	0.83/0.64	0.76/0.36	NG



DRAWING 14

POZZOLANIC AND HYDRAULIC INDEX FOR COMPRESSIVE STRENGTH DEVELOPMENT - ASTM C595

28 DAYS CURING AT 38°C
POZZOLANIC INDEX 35% REPLACEMENT OF PORTLAND CEMENT
HYDRAULIC INDEX 70% REPLACEMENT OF PORTLAND CEMENT

INDEX is the ratio of slag/cement mortar cube compressive strength to cement mortar cube (control) compressive strength.

zolanic index is determined at the 35 percent replacement level while the hydraulic index is determined at the 70 percent replacement level. It is considered that while accelerated curing is involved, the pozzolanic index (Drawing 14) yields valuable information on potential ground non-ferrous slag performance (the hydraulic index, with a much higher replacement level, is not as clear in showing trends and this replacement level is high for ground non-ferrous slags). The ground non-ferrous slags showing "good performance" (PI>0.75) at the 4000 Blaine level were: BDLG; ICNA (excellent for an ambient cooled material); ICNR; KCCR; NRCA (excellent for an ambient cooled material); NRCR; NRCG; and SWLG. The overall good performance of the Noranda non-ferrous slag (NRCA, NRCR, NRCG) should be noted.

TABLE 8 Activation (NaOH) influences on compressive strength development and cement kiln dust replacement of Portland cement tests were, to say the least, a failure and gave poor and scattered results. It is recommended that this avenue of non-ferrous slag study not be pursued at this time.

ANALYSIS AND RECOMMENDATIONS

The eleven non-ferrous slags tested in Phase I (including two foreign slags for comparison, particularly Mount Isa Mines granulated copper slag) are representative of the major types produced in Canada (nickel, copper and lead) and cover the two conventional cooling processes (water granulating and ambient dumping), with possible modifications (vitrification of ambient cooled and lime addition). Grinding, characterization and mortar cube strength development data for the ground non-ferrous slags analyzed in this section can most probably be extended to cover the potential cementitious behaviour of the other Canadian non-ferrous slags: Inco copper slag, Sudbury; Falconbridge nickel-copper slag, Falconbridge; Inco nickel slag, Thompson; Hudson Bay Mining and Smelting copper slag, Flin Flon; and Cominco lead slag, Trail.

Grindability

The grindability of a non-ferrous slag is an important consideration,

TABLE 8

"C109" MORTAR CUBE COMPRESSIVE STRENGTH DEVELOPMENT WITH ACTIVATION
35 AND 70 PERCENT ABSOLUTE VOLUME REPLACEMENT OF PORTLAND CEMENT OR CEMENT KILN DUST
NaOH-SODIUM HYDROXIDE ACTIVATION, 24 HOURS AT 55°C.
CKD-CEMENT KILN DUST SUBSTITUTED FOR PORTLAND CEMENT, 28 DAYS AT 38°C.

		BDLG	ICNA	ICNR	KCCG	KCCR	NMCA
CONTROL		29.6 111	29.6 111	29.6 111	29.6 111	29.6 111	29.6 111
NaOH							
35%							
2000	MPa	5.5	4.4	6.9	5.8	5.5	3.3
3000		6.0	5.1	7.5	7.5	5.9	3.9
4000		9.0	5.9	7.2	7.8	6.8	4.4
5000		11.0	6.7	7.3	NG	6.6	8.2
70%							
2000	MPa	7.7	5.8	4.5	4.2	4.4	2.3
3000		7.8	6.2	5.4	4.9	7.2	3.4
4000		8.2	6.9	6.3	4.5	8.8	5.6
5000		10.5	5.8	7.0	NG	9.3	7.0
PI/HI							
2000		0.19/0.26	0.15/0.20	0.23/0.15	0.20/0.14	0.19/0.15	0.11/0.08
3000		0.20/0.26	0.17/0.21	0.25/0.18	0.25/0.17	0.20/0.24	0.13/0.11
4000		0.30/0.28	0.20/0.23	0.24/0.21	0.26/0.15	0.23/0.28	0.15/0.19
5000		0.37/0.35	0.23/0.20	0.25/0.24	NG	0.22/0.31	0.28/0.24
CKD							
35%							
2000	MPa		-	0.9	-	2.4	
3000			0.8	0.9	-	3.1	
4000			0.8	1.1	-	3.8	
5000			0.8	1.4	NG	5.5	
70%							
2000	MPa		-	0.9	-	4.0	
3000			-	0.6	-	6.2	
4000			-	0.4	-	7.9	
5000			-	0.6	NG	12.3	

NOTES:
1. As for Table 7.
2. Cement kiln dust is from a stockpile (St. Lawrence Cement) so that "aged" material involved. CKD major oxides are: SiO_2 - 17.5%, $A\ell_2O_3$ - 4.9%; Fe_2O_3 - 2.7%; CaO - 39.5%; MgO - 1.9%; Na_2O - 0.5%; K_2O - 5.9%; and LOI - 25.2%.

TABLE 8 Cont.

"C109" MORTAR CUBE COMPRESSIVE STRENGTH DEVELOPMENTS WITH ACTIVATION 35 AND 70 PERCENT ABSOLUTE VOLUME REPLACEMENT OF PORTLAND CEMENT OR CEMENT KILN DUST

		NRCA	NRCR	NRCG	MICG	SWLG
CONTROL		29.6 111	29.6 111	29.6 111	29.6 111	29.6 111
NaOH						
35%						
2000	MPa	4.2	6.0	5.6	6.3	6.1
3000		5.4	7.8	7.3	7.2	5.8
4000		5.8	7.8	8.0	7.0	7.2
5000		6.9	7.7	8.0	8.0	NG
70%						
2000	MPa	4.4	3.9	5.4	3.4	6.4
3000		4.9	5.8	6.5	5.1	8.5
4000		3.0	7.1	6.5	5.0	8.2
5000		2.9	5.2	7.7	7.0	NG
PI/HI						
2000		0.14/0.15	0.20/0.13	0.19/0.18	0.21/0.11	0.21/0.22
3000		0.18/0.17	0.26/0.19	0.25/0.22	0.24/0.17	0.20/0.29
4000		0.20/0.10	0.26/0.24	0.27/0.22	0.24/0.17	0.24/0.28
5000		0.23/0.10	0.26/0.18	0.27/0.26	0.27/0.24	NG
CKD						
35%						
2000	MPa	-	0.7			
3000		-	0.7			
4000		-	0.8			
5000		-	0.8			
70%						
2000	MPa	-	0.3			
3000		-	0.5			
4000		-	0.3			
5000		-	0.2			

given the technical literature indication that a fairly high fineness (Blaine) is required for cementitious applications, and energy (grinding time) requirements are greater, compared to Portland cement clinker. Also, potential use of ground non-ferrous slag requires production level grinding energy information for mill sizing. To consider grindability trends, the energy requirement for 4000 Blaine (selected as reasonable for strength development from mortar cube testing and technical literature, Appendix B-MICG for instance) for each non-ferrous slag can be listed in increasing order with a comparison to initial minus 75 μm, major oxides and degree of vitrification, from Tables 3 and 4 and Drawing 1:

	SLAG	kWh/t	-75 μm, %	SiO_2, %	Fe_2O_3, %	Vitrification
1.	ICNA	46	8	35.7	53.0	crystalline
2.	ICNR	48	1	37.2	53.1	glassy/crystalline
3.	NMCA	76	2	36.3	49.0	crystalline
4.	NRCA	76	3	34.5	49.5	crystalline
5.	NRCG	76	1	34.4	49.6	glassy/crystalline
6.	KCCR	84	.9	22.1	44.4	glassy/crystalline
7.	NRCR	96	.5	36.8	50.0	very glassy
8.	MICG	96	4	42.8	45.8	glassy
9.	SWLG (3000 Blaine 96)	150	.3	27.2	28.5	very glassy
10.	KCCG (3000 Blaine 78)	240	.2	26.5	60.1	very glassy
11.	BDLG (3000 Blaine 72)	261	2	19.9	45.4	very glassy

(CECLG-44 kWh/t, DOPS-48 kWh/t)

From this data summary, it would appear that three factors contribute to decreased grindability - increasing vitrification, decreasing SiO_2 content and decreasing Fe_2O_3 content (total iron expressed as Fe_2O_3) - with degree of vitrification predominating, i.e. glassy slags of even high Fe_2O_3 content such as KCCG "difficult" to grind. Initial minus 75 μm content does not appear to be a factor. It should be noted that there is generally a significant increase in grinding energy in going from 4000 to 5000 Blaine.

Chemistry and Composition

Non-ferrous slags are essentially iron silicates (SiO_2 and Fe_2O_3) with only CaO a significant oxide for lead slags. The determination of glass content (vitrification) was by optical microscopy (not successful given opaque particles) and qualitative X-Ray diffraction (quantitative requires calibration). Clearly, a better method is required given the importance of glass content to grindability and strength development, and the CANMET image analysis method should be adopted.

Mortar Cube Strength Development

For practical applications of ground non-ferrous slag in cementitious applications such as cemented mine backfill and concrete, it is considered that the 4000 Blaine Table 6 and Drawing 13 C989 index at 28 days of 23°C curing (i.e. close to ambient) and 50 percent Portland cement replacement (i.e. practical maximum usage level anticipated) gives the best indication of potential. The C595 pozzolanic index determined for 38°C curing and 35 percent replacement (accelerated curing and low replacement level) and C595 hydraulic index determined for 38°C curing and 70 percent replacement (accelerated curing and high replacement level) are not considered as significant or appropriate. The 28 day C989 index for 4000 Blaine mortar cubes can be listed in increasing order along with pozzolanic index, hydraulic index, 90 day strength (23°C curing) and degree of vitrification from Tables 4, 6 and 7:

	SLAG	C989 28 Day Index	C595 Pozzolanic Index	C595 Hydraulic Index	90 Day Strength, MPa	Vitrification
1.	ICNA	0.48	0.83	0.58	28.3	crystalline
2.	NMCA	0.52	0.70	0.47	23.2	crystalline
3.	NRCA	0.56	0.85	0.59	29.2	crystalline
4.	NRCG	0.59	0.79	0.50	29.1	glassy/crystalline
5.	ICNR	0.59	0.78	0.44	26.6	glassy/crystalline
6.	KCCR	0.67	0.76	0.45	31.1	glassy/crystalline
7.	MICG	0.73	0.70	0.32	34.8	glassy
	(3000 Blaine 0.54)				(24.9)	
8.	KCCG	0.73	0.69	0.31	31.4	Very glassy
	(3000 Blaine 0.70)				(29.7)	

	SLAG	C989 28 Day Index	C595 Pozzolanic Index	C595 Hydraulic Index	90 Day Strength, MPa	Vitrification
9.	BDLG	0.81	0.96	0.29	40.2	very glassy
	(3000 Blaine 0.64)				(37.9)	
10.	NRCR	0.83	0.80	0.54	37.3	very glassy
	(3000 Blaine 0.67)				(32.0)	
11.	SWLG	0.91	0.79	0.37	43.2	very glassy
	(3000 Blaine 0.70)				(39.6)	

From this data, it is clear that there is potential for cementitious use at the 50 percent Portland cement replacement level for MICG, KCCG, BDLG, NRCR and SWLG, using a C989 index > 0.70 as the criteria. Strength development generally increases with increasing vitrification, but increased grinding energy/costs will also be involved. A balance between desired strength development and grinding economics is required. For this reason, lower Portland cement replacement levels and/or Blaine for ground non-ferrous slags may be desirable in practice. Lower replacement levels may also permit the use of less glassy, ground non-ferrous slags such as ICNA and NRCA in cementitious applications.

Recommendations

1. The chemistry and composition (glass content) of the other Canadian non-ferrous slags should be determined for comparison with the eleven slags considered. This will give an indication of their potential for cementitious applications.

2. The degree of vitrification (glass content) of the non-ferrous slags should be determined using the CANMET image analysis method. The information is important given the general influence cf glass content on both grindability and strength development, and will allow a more detailed analysis of the data.

3. The ASTM C989 index approach for mortar cubes should be adopted for determining cementitious potential with extension to include: check on filler influence of non-ferrous slags (i.e. Keil index with Portland cement, ground non-ferrous slag and silica flour); lower Portland

cement replacement levels (i.e. 15 to 30 percent range typically adopted for separately ground iron blast furnace slag); and ternary combinations of ground non-ferrous slag, ground iron blast furnace slag and Portland cement (combinations including fly ash may also be of interest).

4. The Phase I strength development results for a number of the non-ferrous slags are considered promising, and the Phase II study on concrete mix designs and testing for six ground non-ferrous slags should be completed. Subject to review with the Scientific Authority, the six slags suggested for Phase II are:

   ICNA (4000 Blaine) - While C989 index low, there are large quantities of this nickel slag produced. Potential local use in cemented backfill (25 percent substitution level may be appropriate).

   NRCA (4000 Blaine) - Same reasons as ICNA.

   NRCR (4000 Blaine) - Good C989 index and granulation possible at smelter. Potential local use in cemented backfill and/or concrete.

   KCCG (3000 Blaine) - Good C989 index for this granulated slag. Potential local use in cemented backfill and/or concrete.

   BDLG (3000 Blaine) - Fair C989 index for this granulated slag. Local use may be limited and lead content an environmental concern.

   MICG (4000 Blaine) - Control non-ferrous slag.

5. While cemented backfill generally involves high water/cement ratios and tailings, the trend is to more "concrete like" mixes. For this reason, and to cover concrete applications, the mixes in Phase II should be conventional concrete mixes. If possible, additional mixes incorporating tailings should be included.

APPENDIX A

GENERAL NON-FERROUS SLAG USES

Abstracted from:

Slags as Industrial Minerals
J. Emery
Third Industrial Minerals International Congress, 1978
Metal Bulletin Group

Further information on Canadian production and utilization
of non-ferrous slags given in
Canadian Ferrous and Non-Ferrous Slags for Resource
and Energy Conservation
E. Douglas
CANMET
ERP/MSL Report No. 83-61 (TR)

NON-FERROUS SLAGS

The non-ferrous slags considered - copper, lead, lead-zinc, nickel, and zinc - are those that appear to be produced in quantities that make potential applications attractive [1-3]. In fact, the many uses for these slags that have been developed often puts them in the general class of byproducts with useful applications, rather than just wastes that must be disposed of.

COPPER SLAG

Copper slags are usually discharged from the furnace at 1,100°C to 1,300°C and when cold have a black colour and stony, or glassy, appearance. As a rule, the specific gravity will vary with the iron content, from a low of 2.8 to as high as 3.8 [4]. A typical slag from a primary copper smelter is dumped in a molten state or granulated. Recycling for extraction of metal values is common. It would appear that much copper slag, particularly from secondary smelting, is highly suitable for sanding blasting. For countries with fairly low production levels, this usage will probably take most of the available copper slag [1]. Another small volume application at the research and development stage is the use of copper slag in slag wool production.

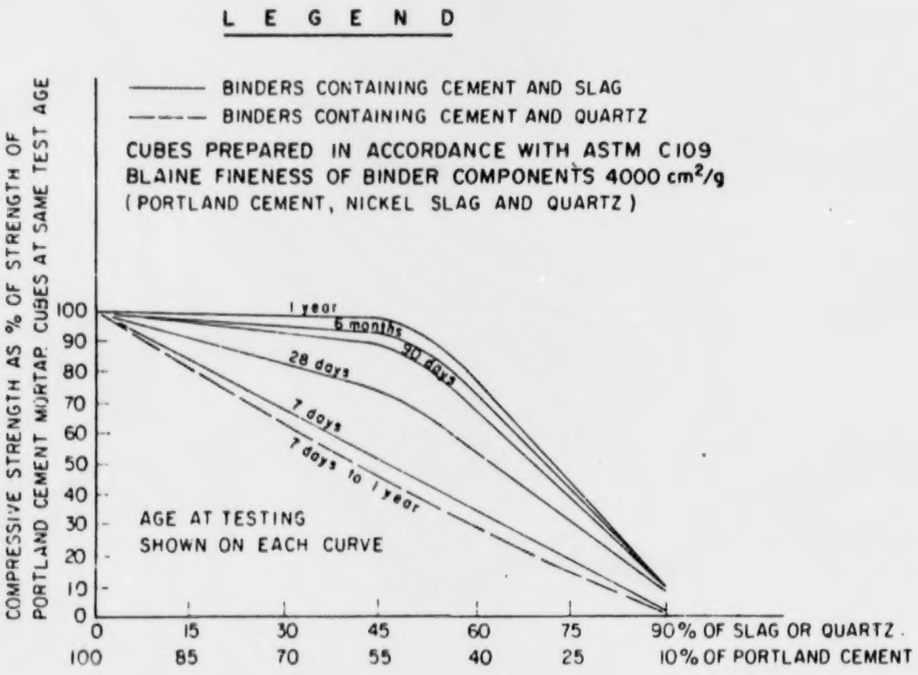


FIGURE 1 Strength of Ground Nickel Slag as a Binder [6].

Current research has shown that several non-ferrous slags (copper, lead, and nickel) have great potential as pozzolans for use in cemented fills, with appreciable cost savings over straight portland cement fill [5, 6]. This is currently done at the Mount Isa mine in Australia and is the subject of research work by several mining companies in Canada. It has also been suggested that a similar cementitious blend of finely ground copper or nickel slag and portland cement could be used as the binder for base stabilization in road construction. Research indicates that the reaction is of a pozzolanic type as indicated by the strength curves for vitrified nickel slag in Figure 1. A summary of the available information on ground non-ferrous slag (copper, lead, and nickel) - portland cement binders indicates the following: must be vitrified; must be finely ground (about portland cement consistency); slag content of the binder must not be too high or reactivity is limited; and the slags act as pozzolans with a relatively slow reactivity.

In areas where large amounts of copper slag are available, use as railway ballast is common [7, 8]. For instance, in Canada copper slag from Noranda has been used with good success as railway ballast since 1968. The Noranda slag has a specific gravity of 3.41 and meets the general requirements for rail ballast. The high unit weight of the slag means that haulage on a volume basis is more expensive than for rock ballast.

It would appear that the only significant current use of copper slag in highway construction is as granular fill and base material. This usually involves crushing and sizing to ensure adequate compaction. As with most non-ferrous slags, potential leachate problems should be checked for before use. The main reason why more copper slag is not used as granular fill and base is the often remote location of copper production from highway construction, and the high unit weight of copper slag. This is particularly the case in Canada and the United States where large volumes of copper slag are generated.

LEAD, LEAD-ZINC, AND ZINC SLAGS

Since lead and zinc are often related as co-products in both source and metallurgical treatment, the various combinations of these non-ferrous slags - lead, lead-zinc and zinc - will be considered as one group. Slags of this group are often black or red in colour, cohesionless, glassy, and have sharp angular particles that are cubical in shape. The specific gravity tends to be as high as 3.6. Slag generation is highly dependent on specific processes and sources, and generally speaking, the quantities available are relatively small.

Lead, lead-zinc, and zinc slags should generally be usable in fill construction and as base materials. However, they are often produced at some distance from construction projects and their high unit weights increase transportation costs on a volume basis. Again, potential leachate and environmental factors must be considered.

NICKEL SLAG

The use of nickel slags in highway and railway construction has been the subject of a fair amount of research in Canada [7-9]. The iron-silicate slag is dumped, and allowed to air cool so that a "clinker" is formed. This material is then crushed and processed as required, but the quantity of fines produced tends to be low. The specific gravity tends to be as high as 3.5. Current annual nickel slag production in Canada is approximately 950,000 tonnes.

Nickel slag from Sudbury has been used for 40 years as railway ballast and has established itself as being the best material used. The processed slag is heavy (specific gravity of 3.2 to 3.6), tough, hard, angular, and packs well under ties to form a better support for track than rock ballast. Slag placed in track over 20 years ago is still in good condition. However, the density of the slag means that transportation on a volume basis is more expensive than rock ballast. More energy is required to compact the material into the track bed, a property that makes the material valuable once in place. This slag meets or exceeds typical specifications for rock ballast. The other applications for nickel slag are similar to those discussed for copper slag above.

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8. Feasby, D.G., "Mineral Wastes as Railway Ballast". Mineral Sciences Laboratory Report MRP/MSL 75-76 (OP), CANMET, EMR, Ottawa, April 1975.

9. Wrong, G.A.,"Experience with Inco Slag for Highway Construction Purposes", Report No. 14, Materials and Research Section, Department of Highways, Ontario, January 1960.

APPENDIX B

INFORMATION RECEIVED FOR SLAGS SUBSEQUENT TO SITE VISITS OR FROM NON-CANADIAN SOURCES

Further information on site visits given in:

Characterization of Non-Ferrous Slags of Canadian Origin
Report on Field Trips Related to the Slag Sampling Program
E. Douglas
CANMET
Report No. MSL-INT 83-4 (FT)

Kidd Creek Mines Ltd. - KCCG, KCT

Mines Noranda Limitée, Murdochville - NMCA

Mines Noranda Limitée, Noranda - NRCA, NRCG, NRT

Mount Isa Mines Limited - MICG

Tsumeb Corporation Limited - SWLG, SWCG

Kidd Creek Mines Ltd. - KCCG, KCT

Kidd Creek Mines Ltd.
B-2002 Timmins, Ontario P4N 7K1
Mine Site (705) 267 2161

FEB - 2 1983

January 20, 1983

Dr. John Emery, P.Eng.
Trow Ltd.
43 Baywood Road,
Rexdale, Ontario.
M9V 3Y8

Dear John;

The samples which you requested have been collected, and are being shipped to you at the above address, C.O.D.

The copper smelter slag was collected over a period of 6 days,(the dates are on the cans), and the chemical analyses are included on the following page.

The sample of tailings was collected in late December. It was decanted three times, then oven dried at 100°C ± 5°C for a period of 24 hours.

I hope that these samples meet your requirements, and do not hesitate to call if you require any further information on the samples, how they were obtained, or any further samples themselves.

Yours truly

David B. Counter

cc: T.R.Yu

KIDD COPPER SMELTER SLAG CHEMICAL ANALYSES

DATE	$\%SiO_2$	%Fe	%S	%MgO	%CaO	$\%Al_2O_3$	$\%Na_2O$	$\%K_2O$
JAN. 5/83	--	45.4	.38	1.24	1.76	3.59	0.23	0.57
JAN. 6/83	30.1	43.0	.96	1.16	1.76	3.23	0.20	0.24
JAN. 10/83	30.7	43.2	.83	1.58	1.12	3.82	0.15	0.14
JAN. 11/83	32.3	43.5	.69	1.41	1.25	3.50	0.15	0.18
JAN. 12/83	30.3	37.3	.79	1.63	1.18	4.12	0.16	0.34
JAN. 17/83	27.4	44.2	.66	1.44	1.50	3.51	0.16	0.45

At the time this letter was written, Tailings analyses were not in yet. They will be forwarded to you by phone in the near future.

The remaining sample fractions are metals and trace elements.

Kidd Creek Mines Ltd.
B-2002 Timmins, Ontario P4N 7K1
Mine Site (705) 267-2161

February 10, 1983

Dr. John Emery, P.Eng.
Trow Ltd.
43 Baywood Road,
Rexdale, Ontario.
M9V 3Y8

Dear John;

Enclosed you will find a summary of the chemical analyses for the tailings from our mill. These assays are a compilation of 15 months of testing. For obvious reasons, metal values cannot be included.

The sample of tailings which you received was a combination of the A ore tails, and the C ore tails. Since production of the tailings is in the ratio of 3 "A" to 1 "C", this ratio can be applied to the analyses given to arrive at a composite assay for the tailings provided.

I hope that this is satisfactory to you, and if you require any further information, do not hesitate to call or write. I remain,

Yours truly,

David B. Counter
Associate Rock Mechanics Eng.

cc: T.Yu

CHEMICAL ANALYSES OF TAILINGS FROM KIDD CREEK MINES (%)

A ORE TAILINGS

SiO_2	60.0
Na_2O	0.3
K_2O	0.6
CaO	1.6
MgO	2.9
Al_2O_3	7.5
FeO	15.4
Trace elements + Base metals	0.8
L.O.I.	10.9 (7.0 Sulphur, 3.0 CO_2, 1.0 H_2O)
TOTAL	100.0

C ORE TAILINGS

SiO_2	42.0
Na_2O	0.2*
K_2O	0.5*
CaO	0.7
MgO	0.8
Al_2O_3	5.0*
FeO	28.8
Trace elements + Base metals	1.8
L.O.I.	20.2 (14.5 Sulphur, 5.0 CO_2, 0.7 H_2O)
TOTAL	100.0

* No specific analyses were available for these elements. The values given are calculated from the A ore tailings analysis, using a downgrading factor due to the extra pyrite content.

MINERALOGICAL COMPOSITION OF KIDD CREEK TAILINGS (%)

A ORE TAILINGS

51.0	Quartz, (SiO_2)
27.0	Silicate minerals, (Chlorite, Sericite, Feldspar)
13.0	Pyrite, (FeS_2)
6.0	Siderite, ($FeCO_3$)
1.5	Other sulphides
98.5	
1.5	Others, (Dolomite, Calcite, etc.)
100.0	

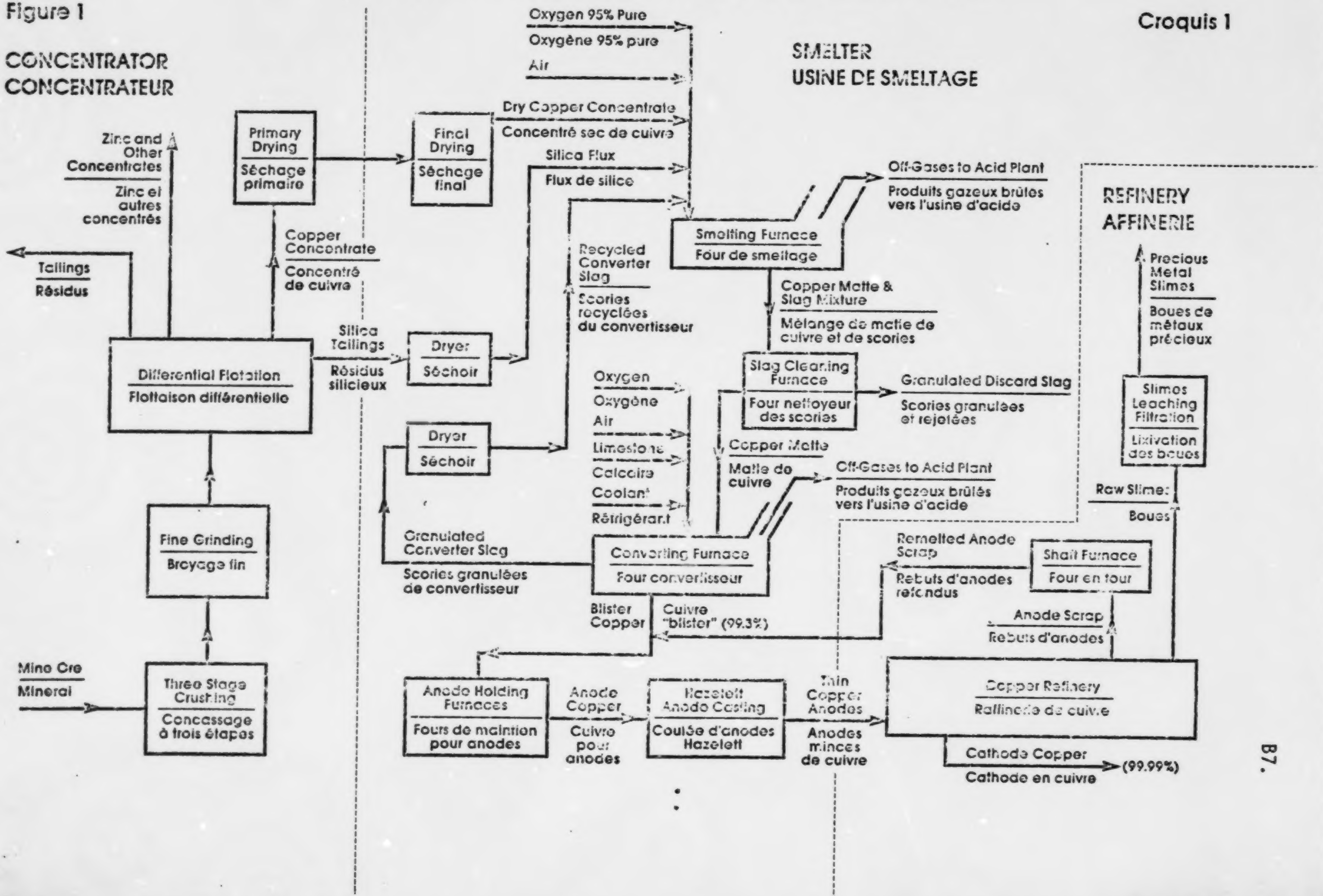
C ORE TAILINGS

36.0	Quartz
18.0	Silicate minerals
28.0	Pyrite
12.0	Siderite
2.0	Other sulphides
96.0	
4.0	Others, (Dolomite, Calcite, Graphite, etc.)
100.0	

FLOW SHEET OF PROCESSES . . . ORE TO CATHODE COPPER
SCHÉMA DES PROCÉDÉS . . . MINERAI À LA CATHODE EN CUIVRE

Figure 1

Croquis 1



Mines Noranda Limitée, Murdochville - NMCA

Tél.: (418) 784-2541

Mines Noranda Limitée
Division **mines gaspé**

Murdochville, Québec G0E 1W0

groupe **noranda**

December 22, 1982

JAN 5 1983

Mr. John T. Emery, Ph.D., P. Eng.
Manager, Advanced Technical Services
Trow Limited, Consulting Engineers
43 Baywood Road
Rexdale, ONTARIO
M9V 3Y8

Dear Mr. Emery:

As per your request dated November 18, 1982 and our telephone conversation following this letter, we have gathered a drumful (about 900 lbs) of fresh reverb slag which has been shipped, collect, to you separately.

Enclosed with this letter are recent monthly composite assays of our reverb slags, as well as a process flowsheet. I hope these are of help to you and that your tests prove successful.

Sincerely yours,

G. Kachaniwsky,
Ass't Smelter Superintendent

/b

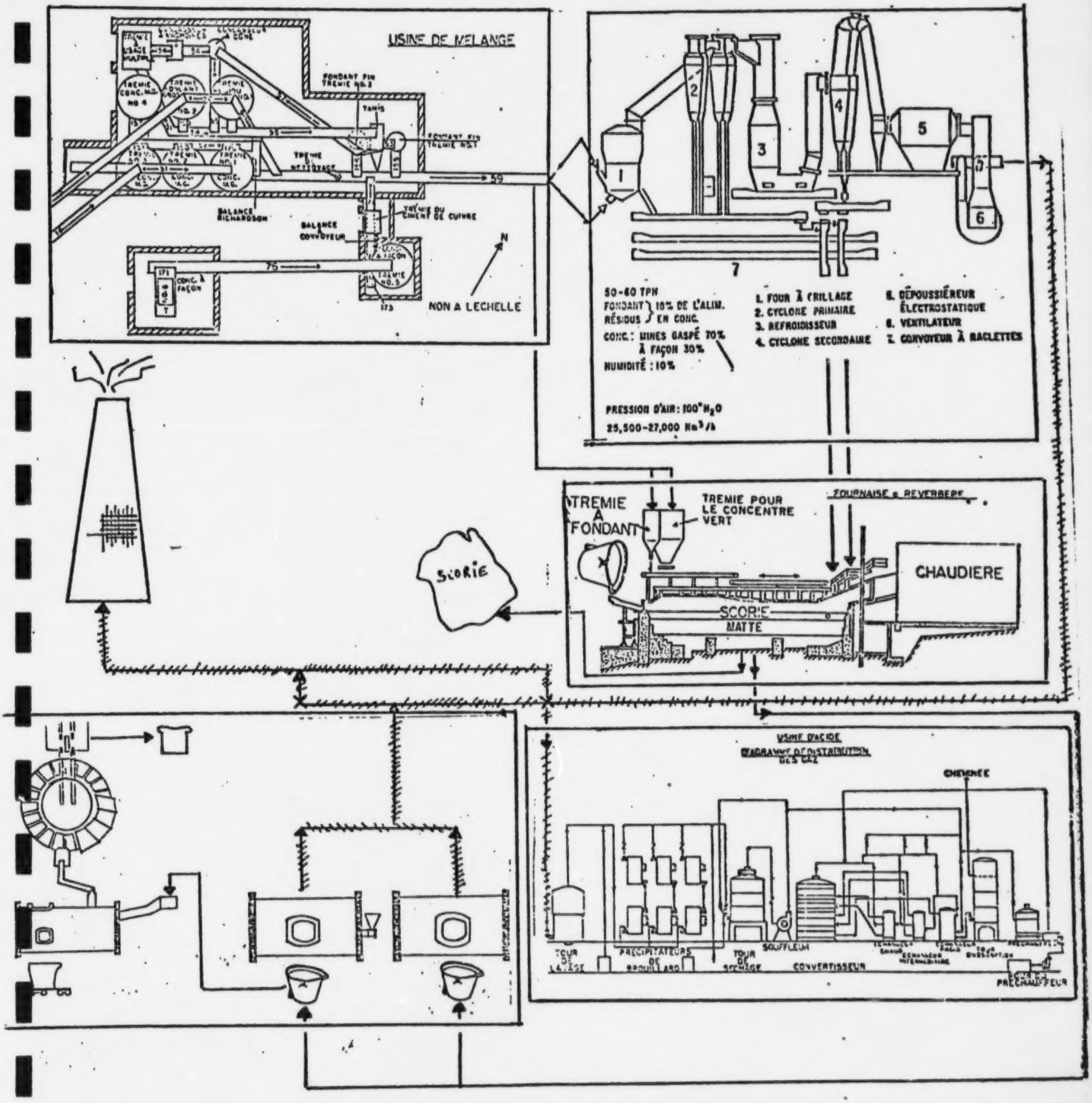
Encl.

c.c.'s: R. R. Faucher
D. G. Pannell

Rev. Slag Analysis

	Au.	Ag.	Cu%	SiO_2	Al_2O_3	Fe	MgO	CaO	S	Bi	Pb	Zn	As	Fe_3O_4	Sb
Aug.	.0022	.12	.432	36.04	2.05	31.60	1.42	6.37	.66	.010	.61	1.37	.030	9.71	.06
Sept.	.0007	.07	.350	36.85	1.94	32.00	1.50	5.95	.66	.008	.62	1.60	.065		
Oct.	.0018	.120	.346	37.35	1.57	32.40	1.33	5.60	.65	.005	.53	2.14	.080		.08
Nov.	.0021	.190	.315	37.95	[illegible]	32.60	1.50	[illegible]	.57		.89	2.35			

USINE DE SMELTAGE



Mines Noranda Limitée, Noranda - NRCA, NRCG, NRT

Mines Noranda Limitée
Division Horne
C.P. 4000
Noranda, Qué. J9X 5B6

noranda

JAN 11 1983

January 6th, 1983

Dr. J.J. Emery, Manager
Advanced Technical Services
Trow Limited
43 Baywood Road
REXDALE, Ontario
M9V 3Y8

Dear John:

The following slag samples have been shipped, to your attention, in 45 gallon drums.

Sample A:	2, 45 gallon drums of reverberatory slag.
Sample B:	2, 45 gallon drums of granulated reverberatory slag.
Sample C:	Noranda Reactor slag tailings - 100 lbs.

A representative sample of each was taken for chemical analysis which follows:

	Fe, %	SiO_2, %	Cu, %
Sample A	34.3	36.0	0.5
Sample B	34.6	35.3	0.46
Sample C	40.0	24.5	0.39

Background information on source and sampling is given below.

Sample A (Reverberatory Slag)

Reverberatory slag is tapped into ladles and brought to the slag dump where it is poured in the liquid state unto a sloping surface. When cool, it is recovered and transported to the Noranda CN yard where it is crushed and stockpiled. The sample shipped to you was taken half-way up the 50' high stockpile.

Approximately 2 feet of slag was removed from the surface before filling the drums using a shovel from a single location. Samples were taken at regular intervals, while filling the drums, to make up the representative sample for chemical analysis.

This slag sample is representative of the current cooling and handling conditions of our reverberatory slag.

Sample B (Granulated Reverberatory Slag)

This sample was taken from a pile (very approximate 75,000 tons) of slag which was granulated some 8 years back. After removing some slag from the surface (top of the pile) the drums were filled using a shovel from a single location. A representative sample was taken for chemical analysis.

Sample C

Slag from the Noranda reactor is slow cooled, milled and treated by a flotation process to remove the copper content and generate a disposable slag tailing. Representative slag tailing samples are sent to the analytical laboratory on an 8 hour basis. These samples were collected from a 1-month period to make up Sample C.

I hope these samples and the corresponding information are sufficient for the study. If not do not hesitate to give me a call.

During your visit at the Horne you briefly described the method of encapsulating dangerous fine materials in concrete. This is an area we are interested and would appreciate receiving information you may have available.

Yours very truly,

D. Poggi
Metallurgical Superintendent

DP/dv

cc L.A. Mills

Mount Isa Mines Limited - MICG

B16.

MOUNT ISA MINES LIMITED
MOUNT ISA OPERATIONS
(A MEMBER OF THE M.I.M. HOLDINGS GROUP OF COMPANIES)

MOUNT ISA, QUEENSLAND 4825, AUSTRALIA
TELEPHONE: (077) 44 2011 MOUNT ISA
TELEGRAMS: "ISA MINE", MOUNT ISA
TELEX: AA 49662

ADDRESS REPLIES TO
GENERAL MANAGER

IN REPLY QUOTE

REF.: GJA/7.5/FIL 1.2

DATE: January 4 1983

Mr John Emery
Trow Ltd
43 Baywood Road
Rexdale
ONTARIO M9V 3Y8

CANADA

JAN 2

COPPER REVERBERATORY FURNACE SLAG

Dear Mr Emery,

In response to your telex of December 14, we have despatched by sea about 200 lb (2 containers) of Copper Reverberatory Furnace Slag. You will notice that they are marked F1 and F2 to distinguish the slags from our two furnaces. You will also notice that they are unground; we specify that they must be ground to a fineness where more than 90% is less than 37 µm.

It was established some time ago that lead slag was more reactive than the copper, but for various reasons it has not been used. Because of this we have not forwarded any to you.

As for the specific information you wanted:

- Dr E.G. Thomas is now in Canada. You should have received a telex informing you of his whereabouts. We also sent him a telex to inform him of your interest. He is lecturing at the University of New South Wales during the academic year.
- All of Dr Thomas' work in this regard has been widely published in mining journals, so if you have done a literature search you have all the papers we could give you. The system at MIM has not changed significantly since its inception. Dr Thomas' papers explain this system.
- There is a study of the chemistry of our cemented fill currently being conducted on our behalf by the Civil and Mining Engineering Department of the University of Sydney. A paper on the progress to date will be presented by Dr H. Roper at the "First International Conference on the Use of Fly Ash, Silica Fume, Slag and Other Mineral By-Products in Concrete", July 31 - August 5 1983, Montebello, Canada. For more general information you could contact Dr Roper in Sydney, or Mr V.M. Malhotra who is the chairman of the organising committee for the conference. His address is:

  Canada Centre for Mineral and Energy Technology
  555 Booth Street
  Ottawa Ontario K1A 0G1

We hope that we have been of assistance to you. Please do not hesitate to contact us if we can help you in any way.

Yours faithfully,

MOUNT ISA MINES LIMITED

For General Manager

B18.

THE UNIVERSITY OF NEW SOUTH WALES

P.O. BOX 1 • KENSINGTON • NEW SOUTH WALES • AUSTRALIA • 2033
TELEX AA26054 • TELEGRAPH: UNITECH, SYDNEY • TELEPHONE 663 0351
EXTN.

PLEASE QUOTE

SCHOOL OF MINING ENGINEERING

APR 11 1983

31 March, 1983

Dr. J.J. Emery
Trow Ltd.
43 Baywood Road
Rexdale
Ontario
Canada M9V 3Y8

Dear John

Please excuse my rather belated reply to your request of early January. Life has been quite hectic in the interim, but I am now back in Sydney with most of the urgent matters under control.

I have had a look through your proposal and cannot offer any criticism of any significance. I would tend to use smaller test samples, but then my interest is more soil mechanics than your concrete development, which requires larger test specimens. I would also question the need to grind to 6000 Blaine and would take 3000 and 4000 as being adequate.

As indicated, neither comment is of great significance. You seem to be embarking on a very interesting and comprehensive study. The real fun will start after you have your (positive) results and you set out to sell your ideas to mining companies and concrete producers.

If you feel I can contribute to the study in any way, formal or informal, paid or unpaid, please let me know.

Kind regards,

Yours sincerely

E.G. Thomas
Senior Lecturer
in Mining Engineering

Tsumeb Corporation Limited - SWLG, SWCG

B20.

TSUMEB CORPORATION LIMITED

(INCORPORATED IN SOUTH WEST AFRICA/NAMIBIA)

Telephone: National (0671) 2100
International +27 671
Telex: 908-56-680 WK
Cables: "Tsumcorp", Tsumeb

P.O. Box 40
TSUMEB
South West Africa/Namibia
9000

When Replying
Please Quote Our Ref. No. S-8

13 January 1983

FEB - 1 1983

Dr. J.J. Emery
Trow Limited
Consulting Engineers
43 Baywood Road
Rexdale
ONTARIO, CANADA
M9V 3Y8

Dear Dr. Emery

SMELTER SLAG SAMPLES FOR CANMET STUDY

We apologize for the delay in getting the above samples despatched to you. Sea freighting was possible via Cape Town only, while air freight depended on possibilities for consolidation of batches.

The samples left South Africa on flight BA54 on January 12, and are due for arrival Toronto on flight BA073, January 14. Airway bill number is 125 30660243. The samples were sent freight pre-paid, and you will be invoiced at a later date when documents become available.

Samples are well granulated slag produced at the Tsumeb smelters under the best granulating conditions of high water flow rate and low slag pour rate, and consist of two lots of 56,8 kg each clean Reverberatory slag and two lots of 56,8 kg each of Lead Blast Furnace slag.

Attached are, as requested:

1. Chemical analyses of the two samples supplied.
2. A mineralogical report on a sample of Reverb slag similar to the one supplied.
3. Flowsheet data to indicate where samples were generated.

2/....

Directors:
R. C. Bonebrake, USA, Wayne H. Burt, USA, J. P. Du Cane, UK, R. D. Ellett, USA, C. O. Ensign, USA, J. F. Frawley, USA, P. Gousseland, USA, Plato Malozemoff, USA, H. A. R. Meiring, G. R. Parker, E. Pavitt, A. H. Schmidt-Fellner, USA, H. Simeon, USA, J. E. Thompson, USA, John Towers, USA.

Alternate Directors:
D. J. Christie, USA

Please note that there are two copper circuits in operation, one treating concentrates high in lead, arsenic, cadmium and zinc, the other one treating clean chalcocite-chalcopyritic concentrates. The reverb slag sample originated from the latter circuit.

It is trusted that these products will assist you in the study looking for potential cementitious applications of slags in terms of backfill and concrete. It is also confirmed that our samples can be specifically identified so that your report will not involve restricted information.

Yours very truly
TSUMEB CORPORATION LIMITED

for
H.A.R. MEIRING
GENERAL MANAGER

WJCV/HARM/jms

c.c. Mr. L. Harris
Mr. E. Hamer

Encl. 3:

B22.

TSUMEB CORPORATION LIMITED

(INCORPORATED IN SOUTH WEST AFRICA/NAMIBIA)

Telephone: National (0671)
International -27 671
Telex:
Cables: "Tsumcorp", Tsumeb
When Replying
Please Quote Our Ref. No.

P.O. Box 40
TSUMEB
South West Africa/Namibia
9000

January 1983

Re. Slag Samples for CANMET

Two slag samples were supplied:

1. Granulated Reverb Slag ex No.2 Copper Circuit, 113.6 kg (250 lb)
2. Granulated Lead Blast Furnace Slag, 113.6 kg

A mineralogical report on a sample of Reverb Slag similar to the one supplied is attached.

Chemical Analysis

Element	% by Mass	
	Reverb Slag	Blast Furnace Slag
Cu	0.59	0.33
Pb	0.54	1.91
Zn	1.90	10.4
Fe	32.0	19.2
As	0.90	0.17
Total Sulfur	1.04	0.66
Sulfate Sulfur	<0.05	<0.05
SiO_2	39.6	28.4
CaO	6.6	20.2
Al	2.3	5.0
Mg	2.0	1.9

Directors:
R. C. Bonebrake USA, Wayne H. Burt, USA, J. P. Du Cane, UK, R. D. Ellett, USA, C. O. Ensign, USA, J. F. Frawley, USA,
P. Gousseland USA, Plato Malozemoff, USA, H. A. R. Mering, G. R. Parker, E. Pavitt, A. H. Schmidt-Fellner, USA,
H. Simeon USA, J. E. Thompson, USA, John Towers, USA

To Dr G.R. Mortimore December 16, 1982
from J. Innes

Mineralogical Report No.862

Subject

Mineral composition and crystallinity of No.2 Reverberatory Furnace Slag, Tsumeb

Mineralogy

The slag consists of approximately 85 vol% amorphous glass and of a remaining 15 vol% crystalline phases.

1 - 2 vol% of the slag are free particles up to 5 mm in size of solid solutions of $CuFeS_2$-$CuFe_2S_3$ as well as Cu_2S-Cu_5FeS_4.

The bulk of the slag consists of glass, containing about 2 vol% magnetite, mainly as submicron-sized crystallites, but occasionally as grains up to 30 µm in size. The glass also contains 1 vol% matte as dispersed spheroids, mainly of submicron size but ranging up to 10 µm.

The glass shards are extremely inhomogeneous in composition: some are entirely composed of coarse (100 µm) skeletal calcium-zinc/calcium-magnesium silicates, whilst others are totally vitrified.

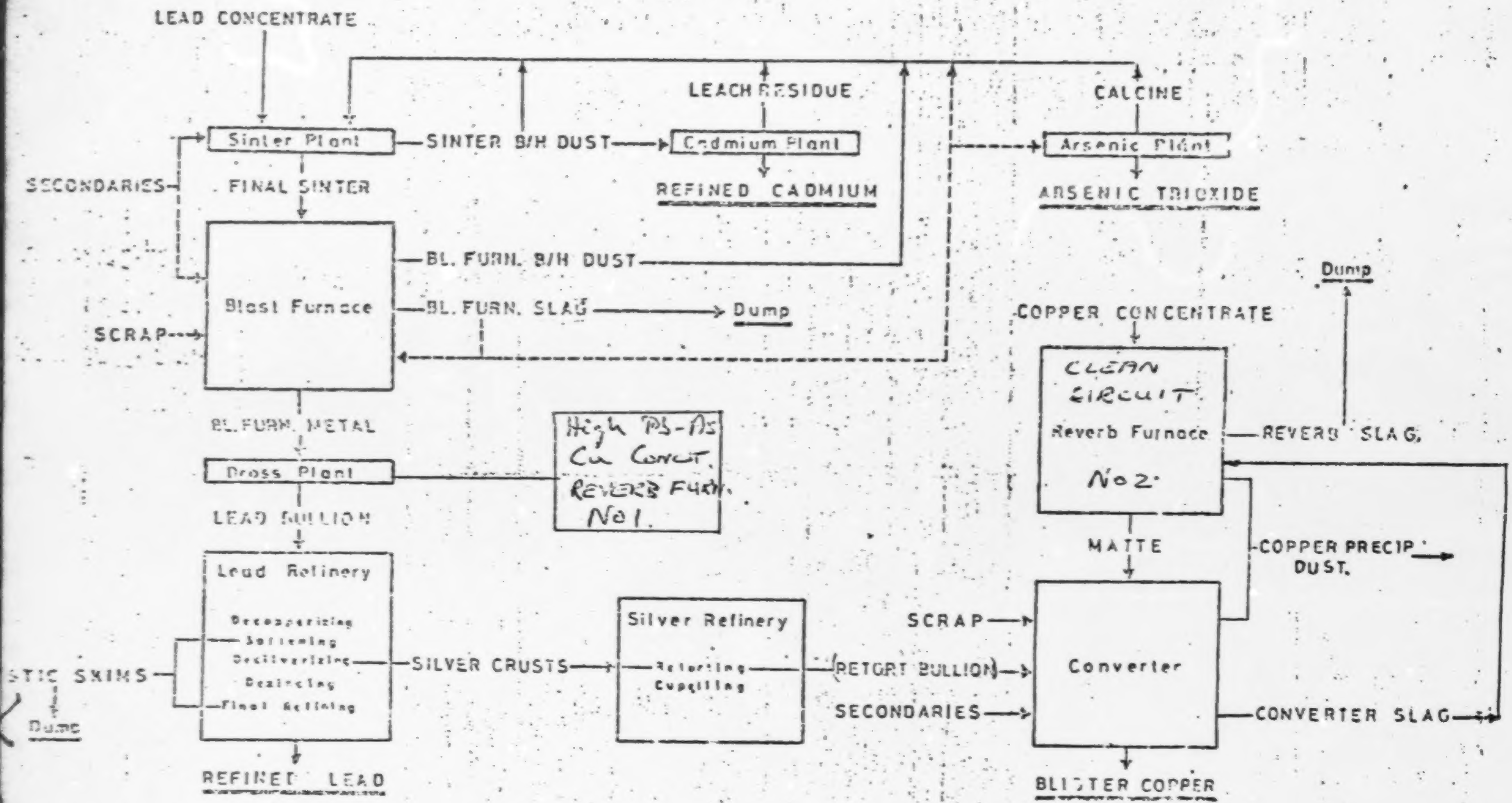
Examination of a grain mount between crossed polarizers indicated that about 10 vol% of the glassy material shows birefringence, implying crystallinity (grain size 10 - 15 µm).

No other phases were recognizable.

J. Innes

c.c.

R.C. Chaplin
File



TSUMEB SMELTER: MATERIAL FLOW OF AND BETWEEN
LEAD & BYPRODUCT SECTION AND NO.2 COPPER PLANT

APPENDIX C
GRINDING CURVES

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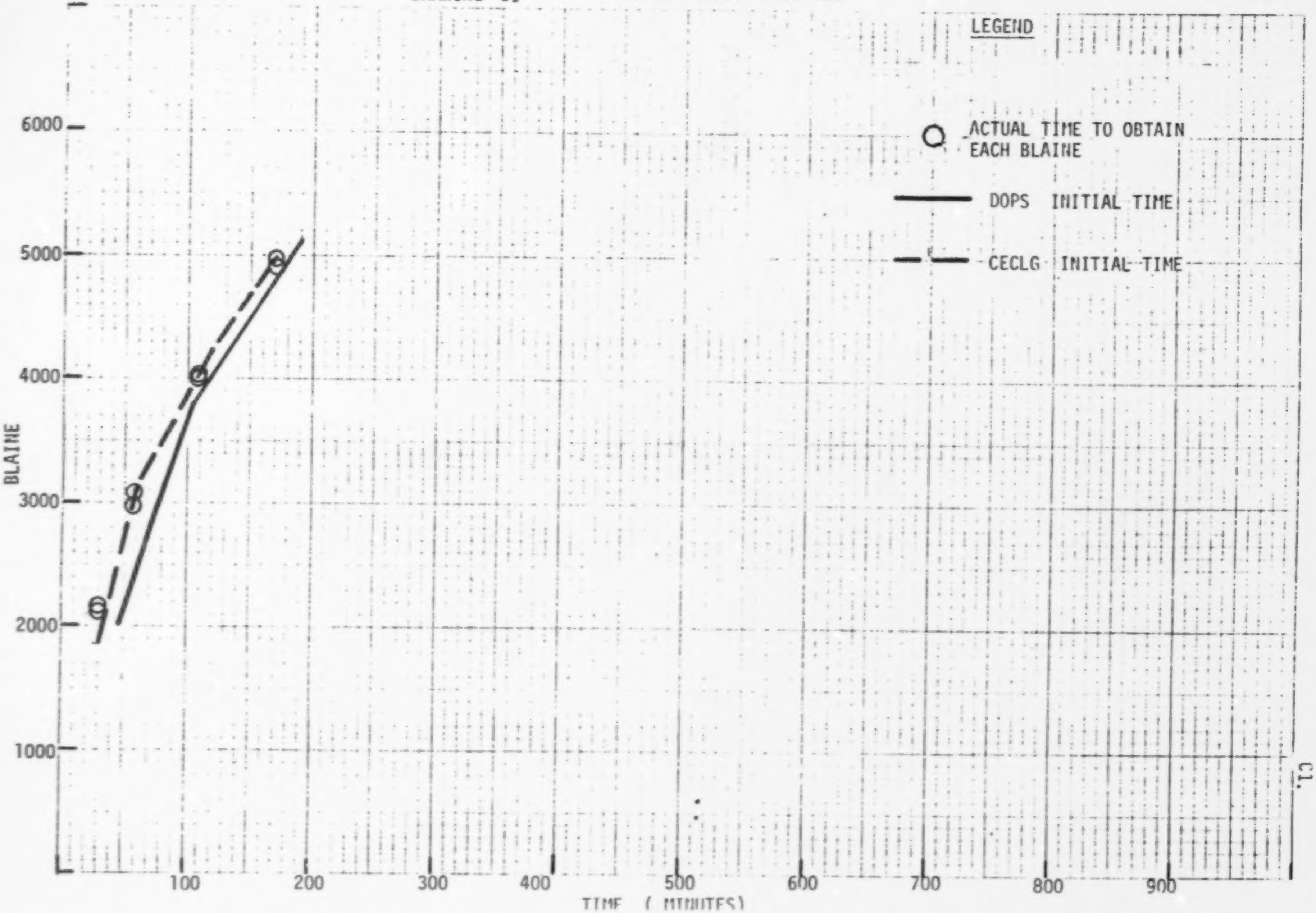

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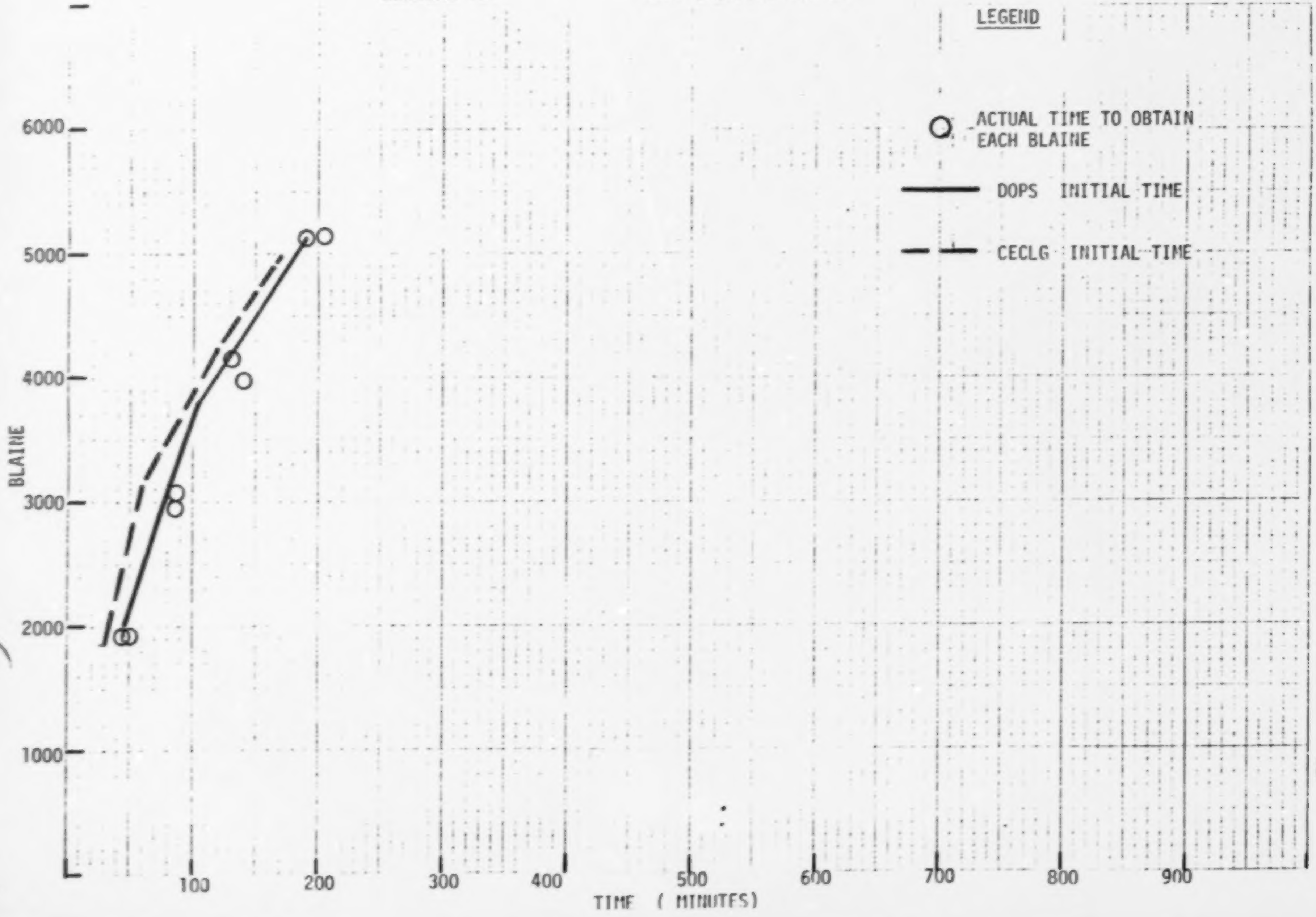

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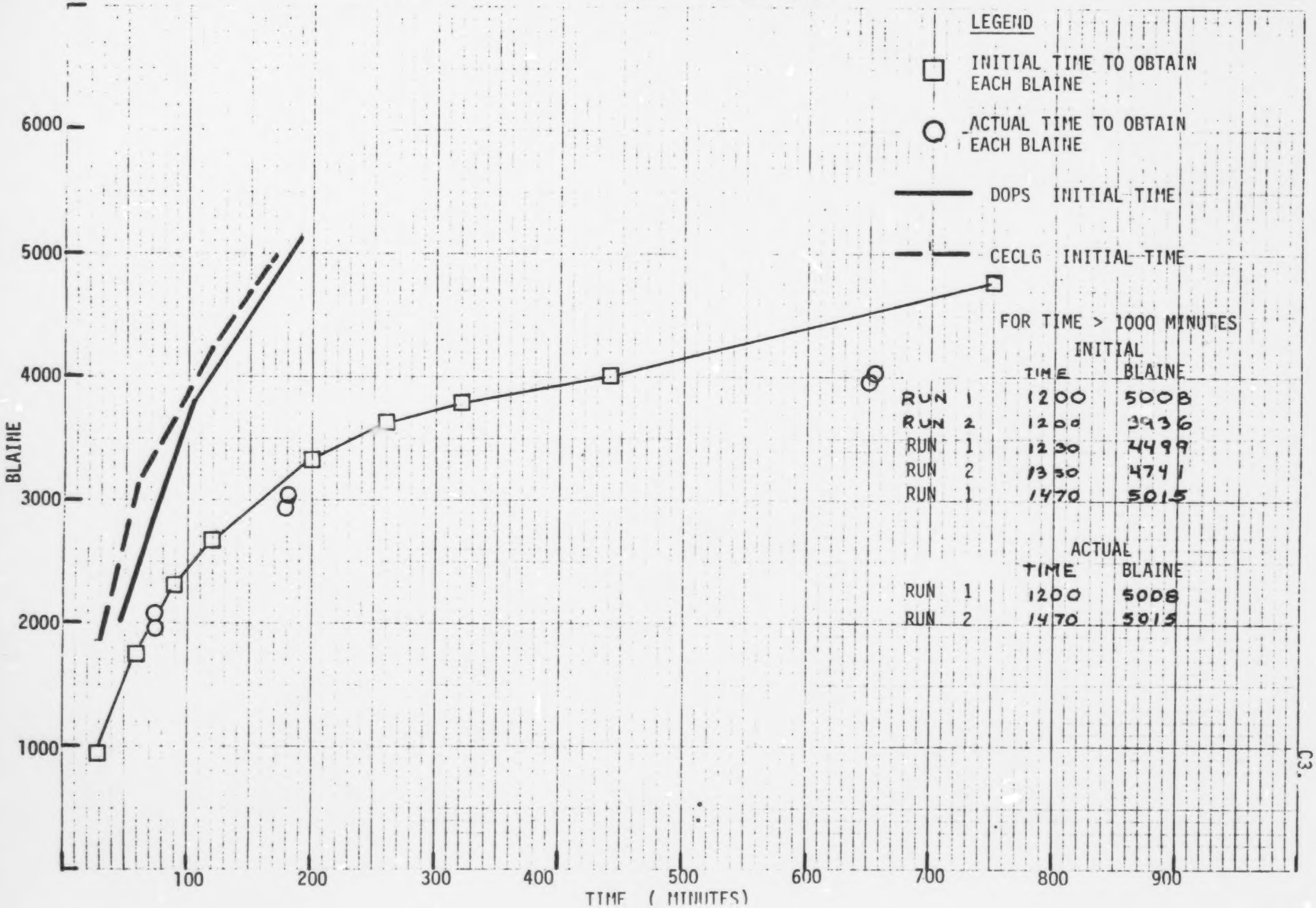

FOR TIME > 1000 MINUTES

INITIAL

	TIME	BLAINE
RUN 1	1200	5008
RUN 2	1200	3936
RUN 1	1230	4499
RUN 2	1350	4741
RUN 1	1470	5015

ACTUAL

	TIME	BLAINE
RUN 1	1200	5008
RUN 2	1470	5015

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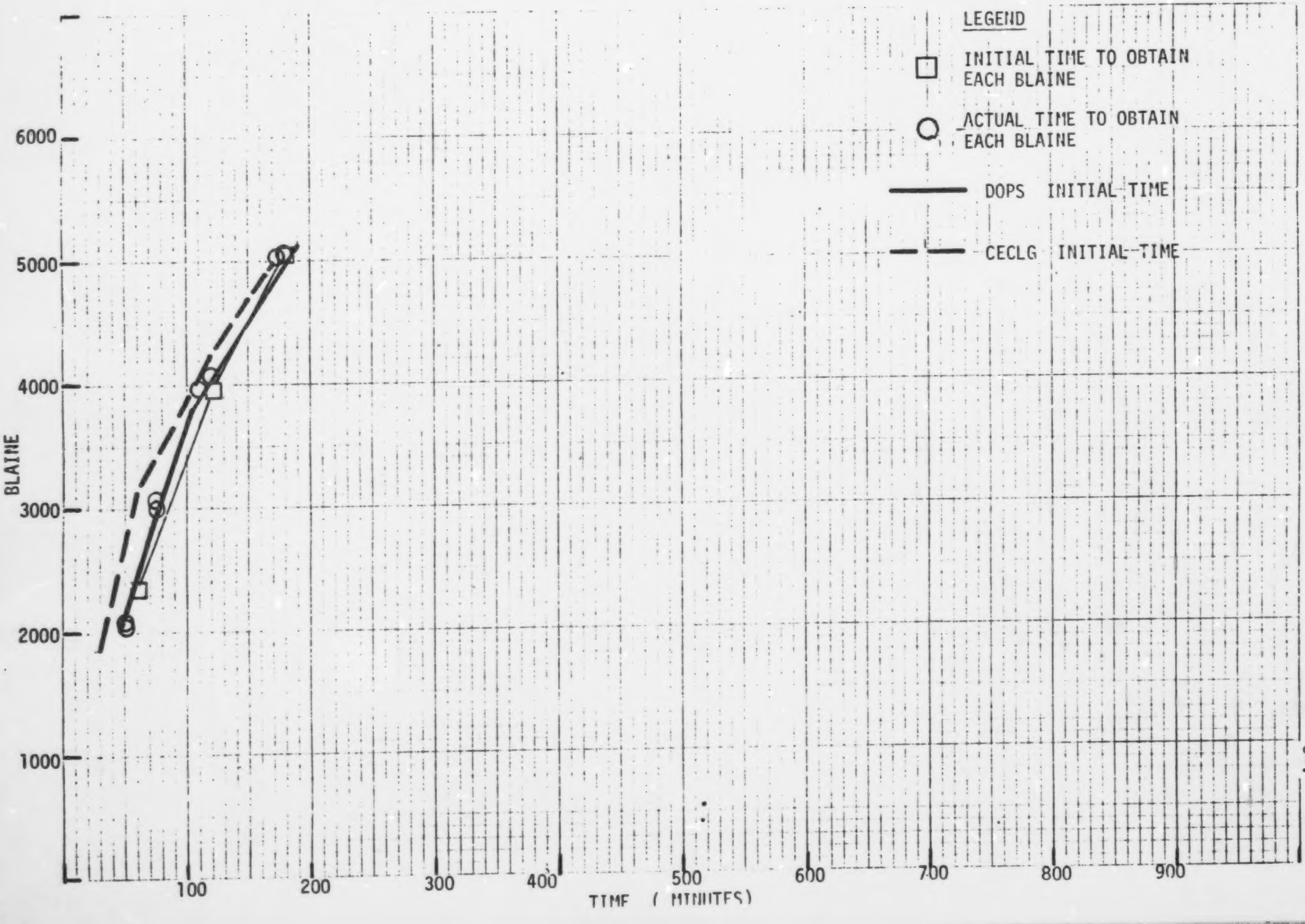

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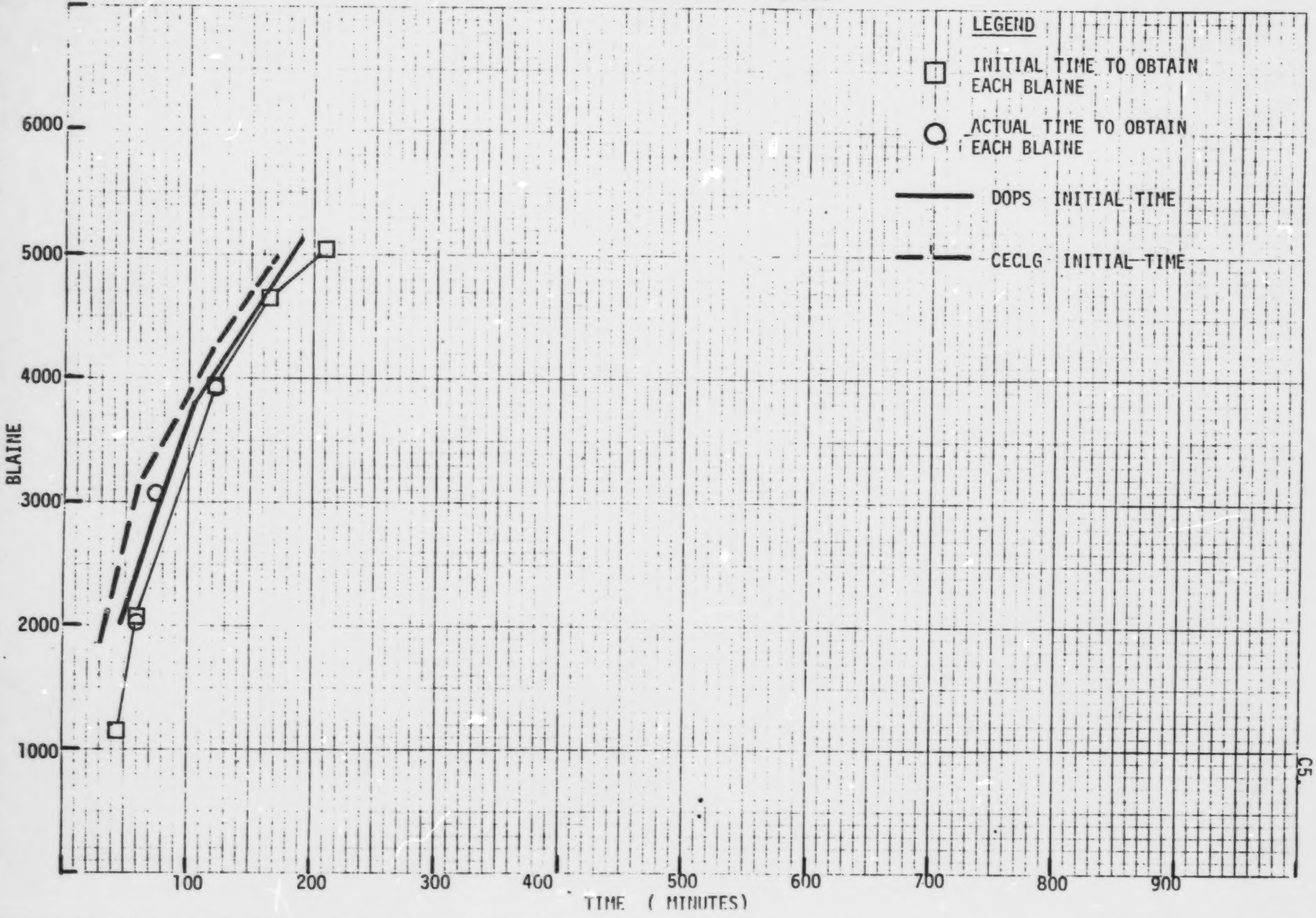

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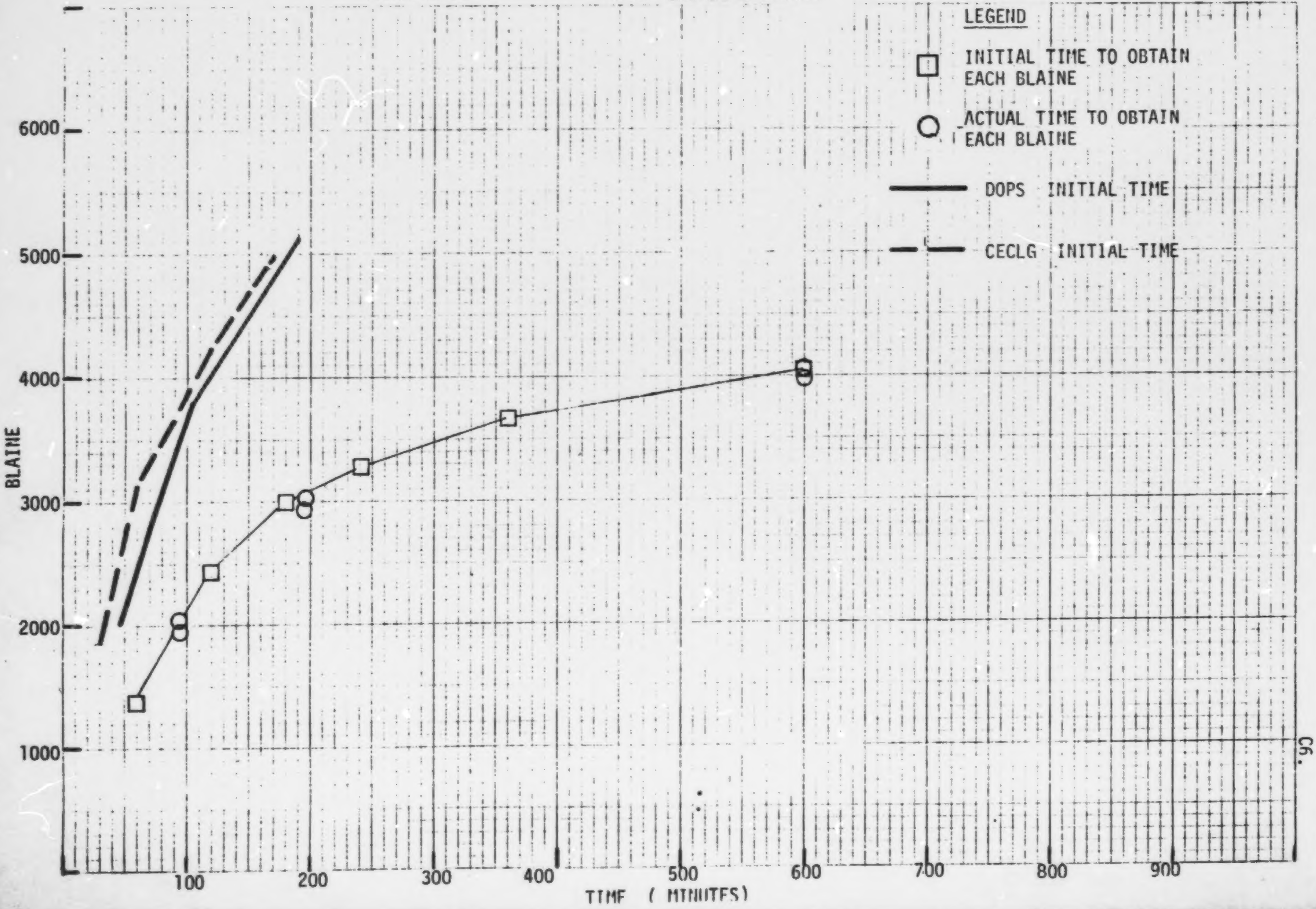

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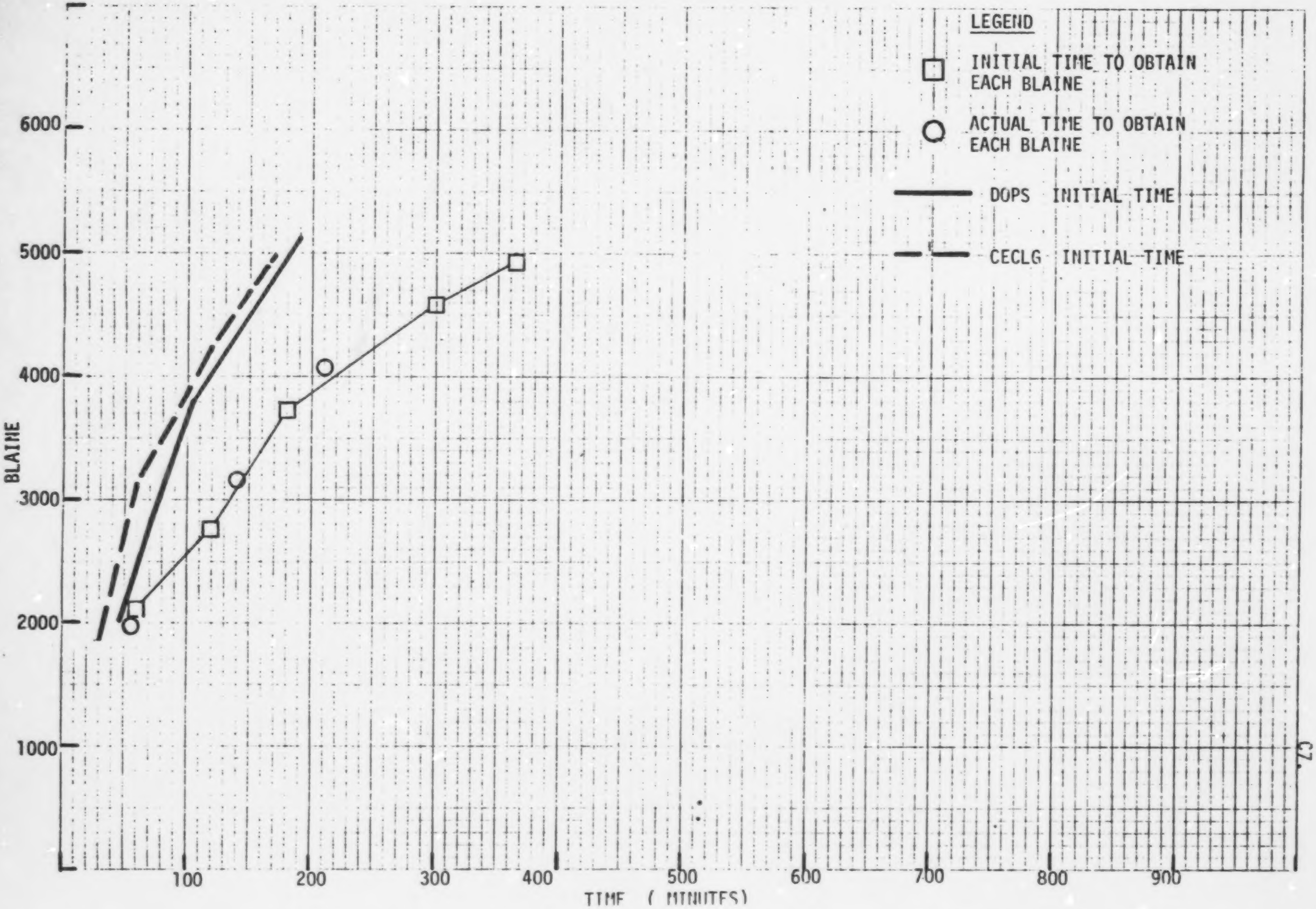

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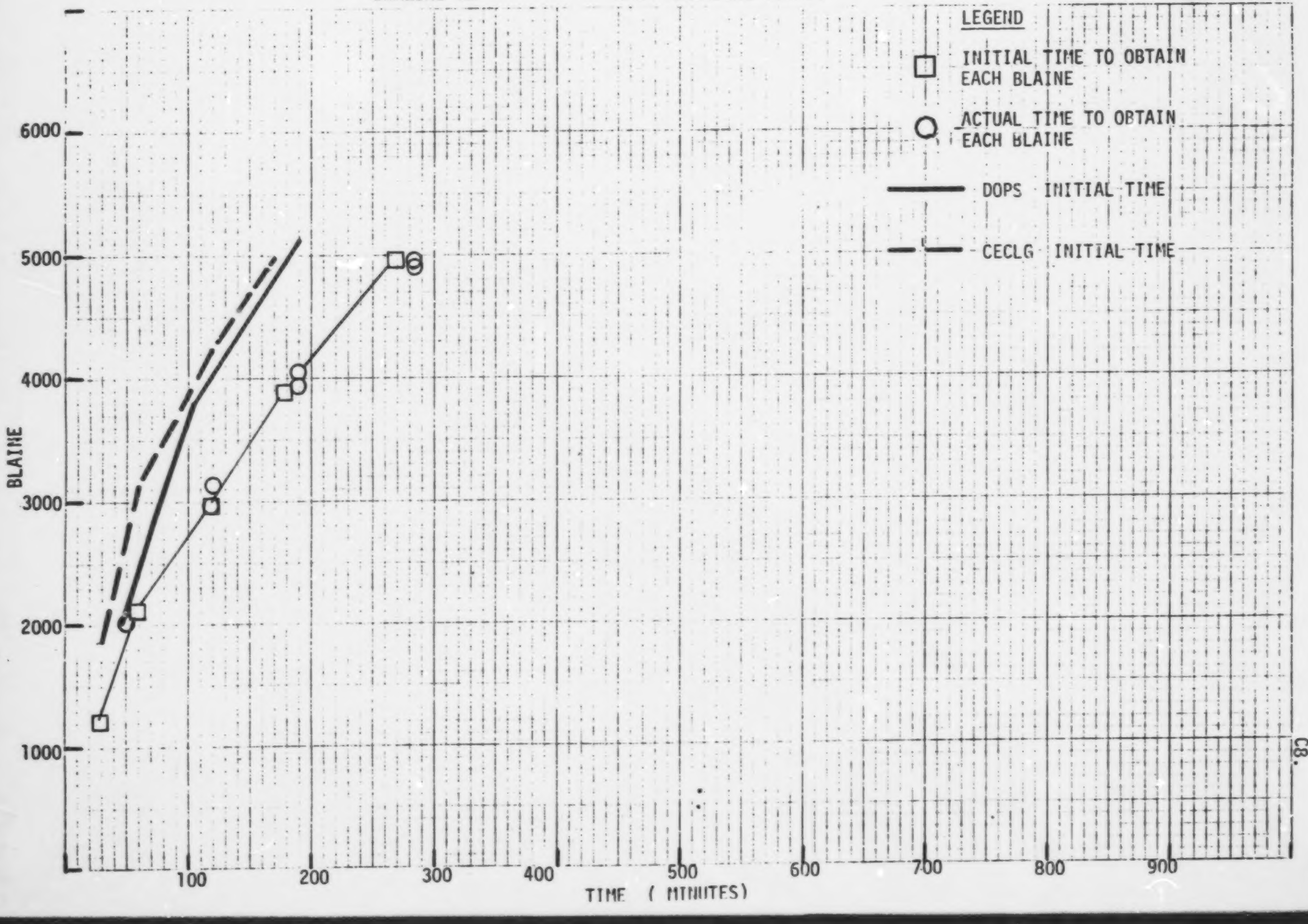

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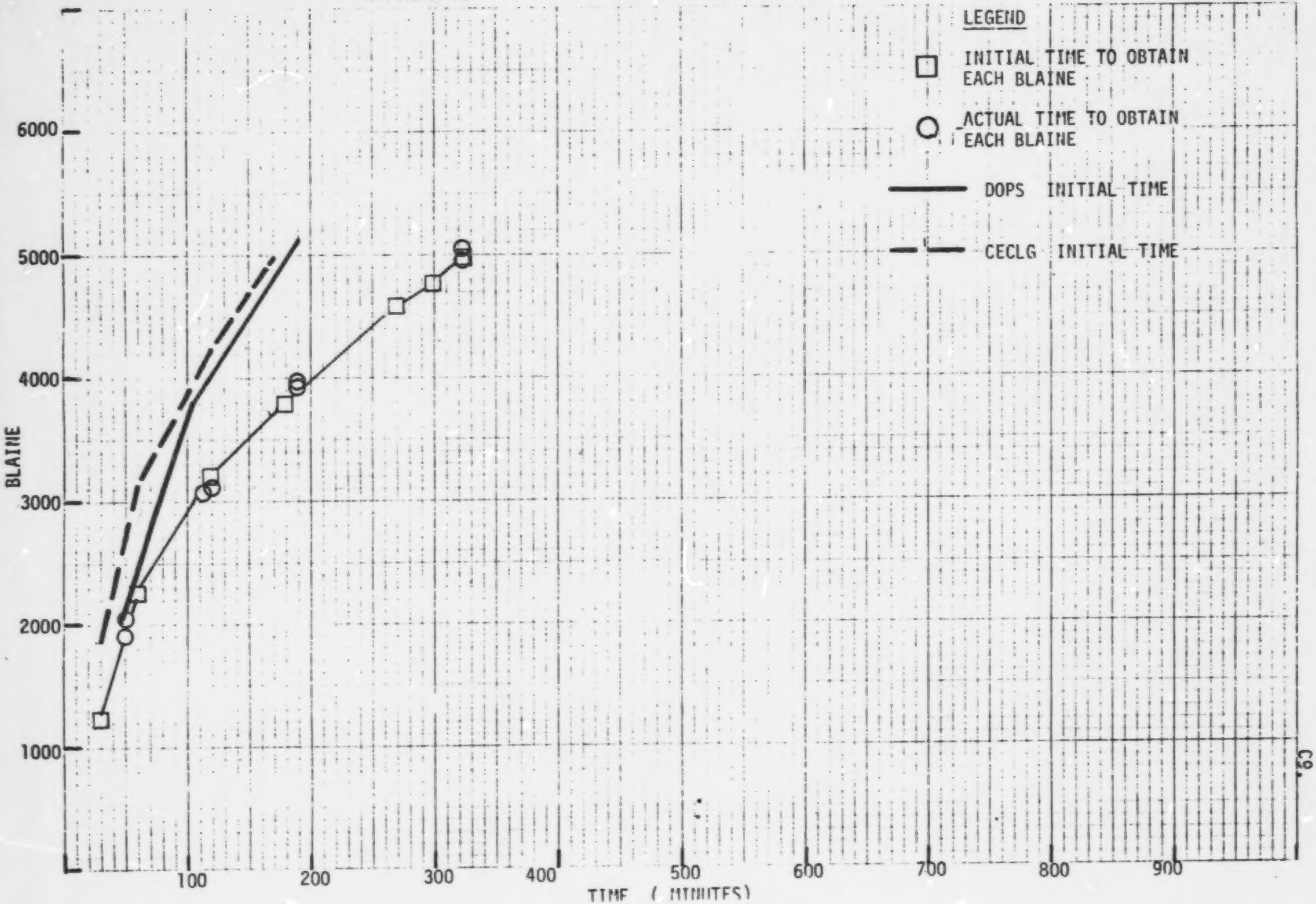

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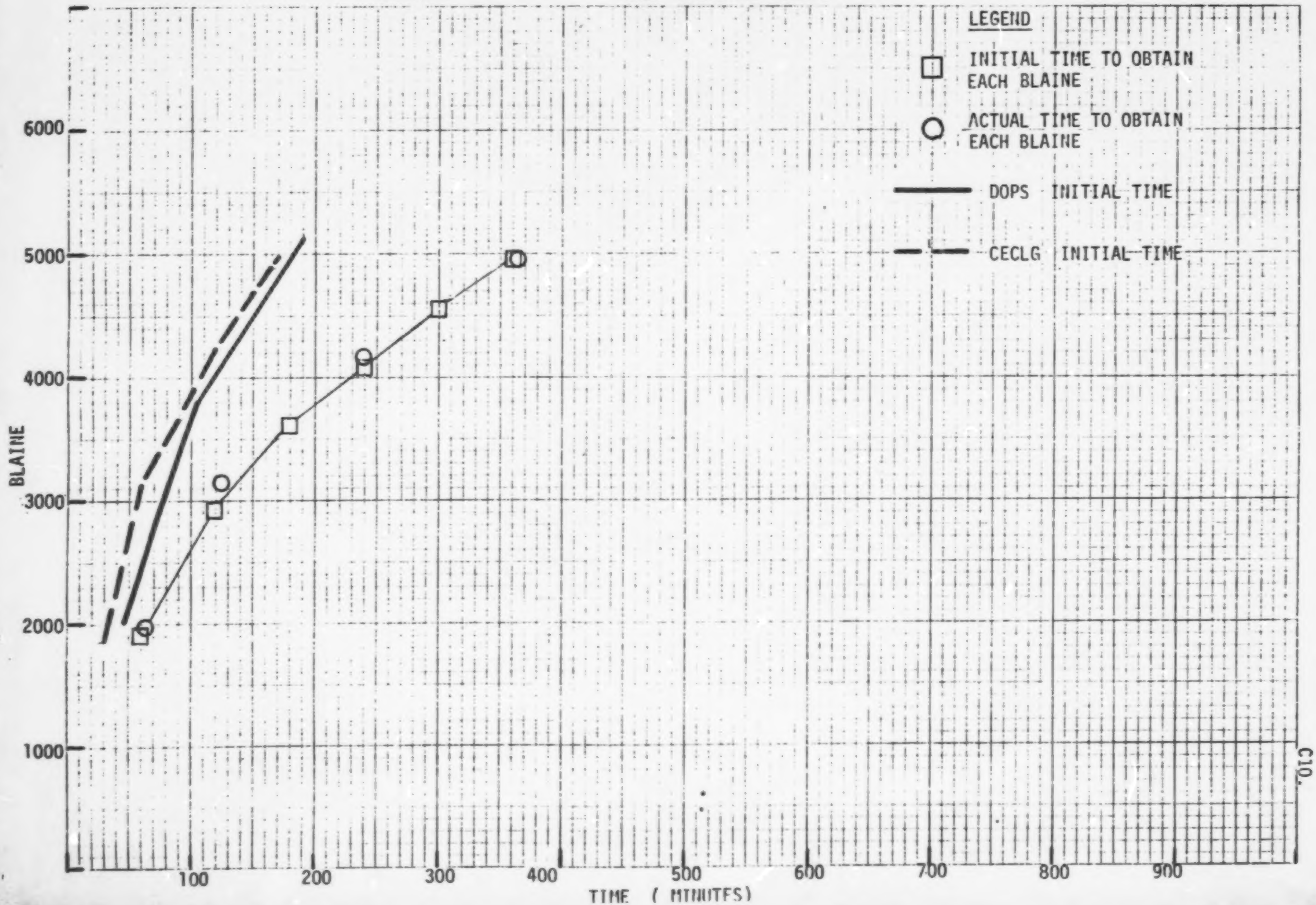

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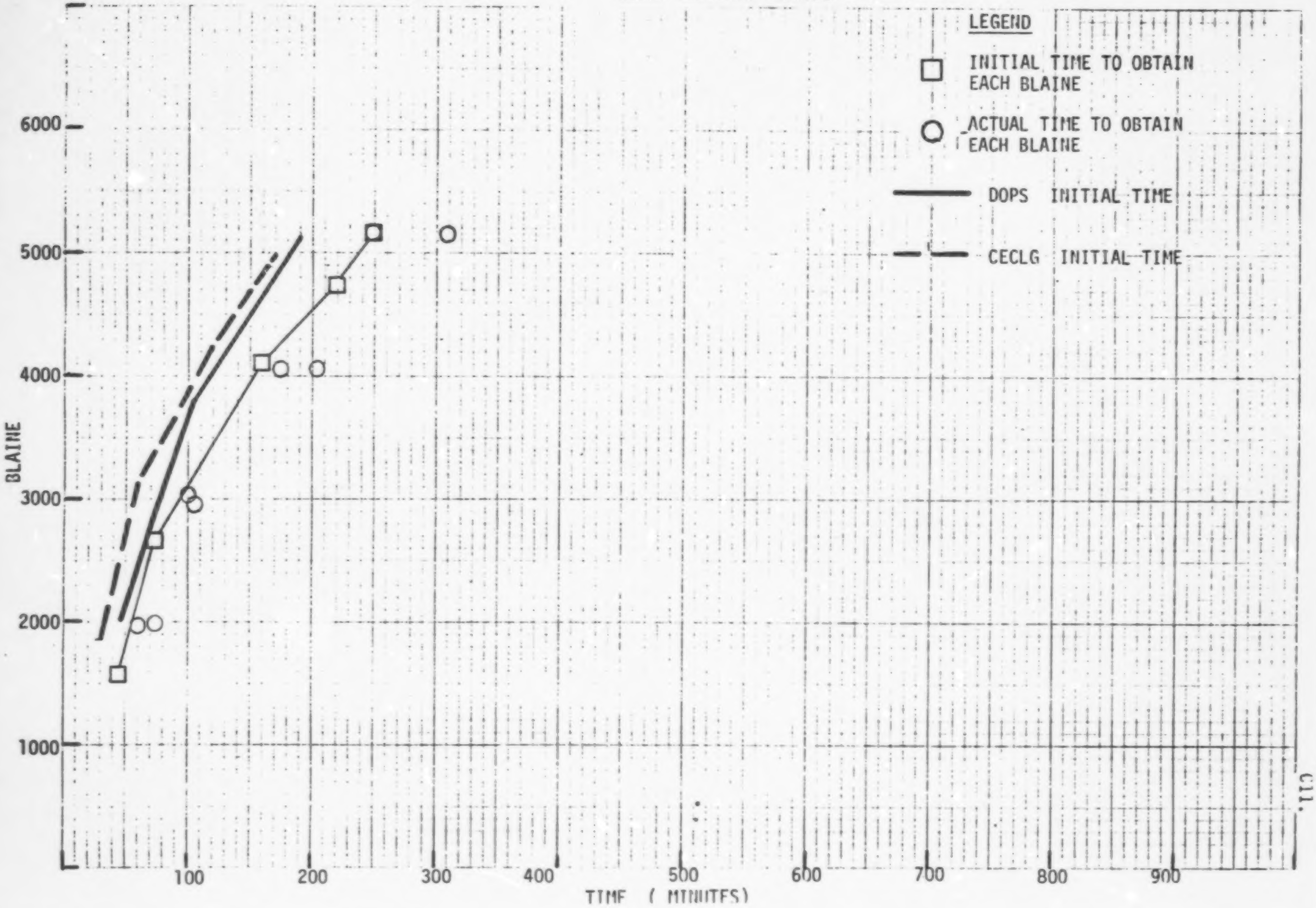

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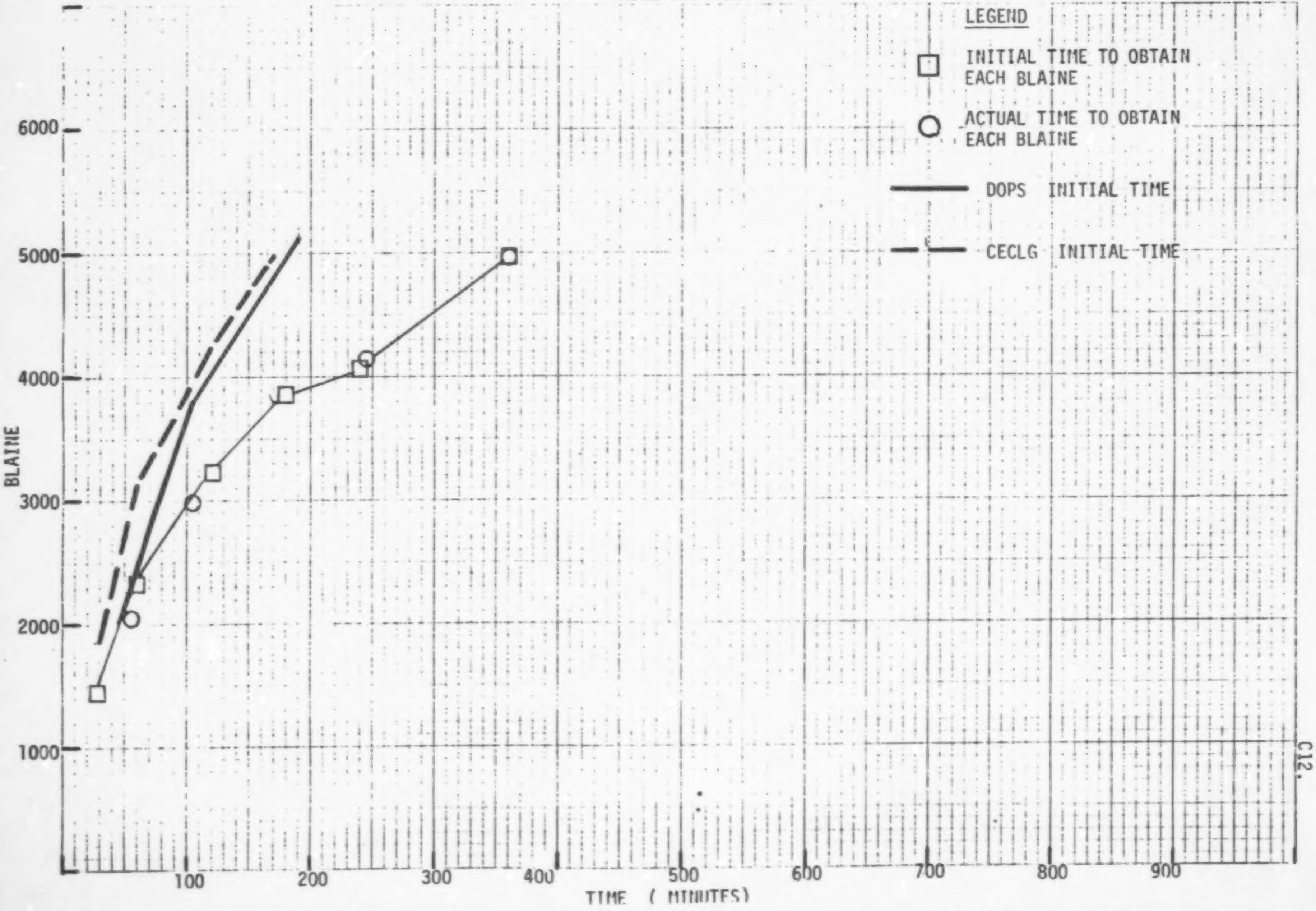
MICG GRINDING CURVES



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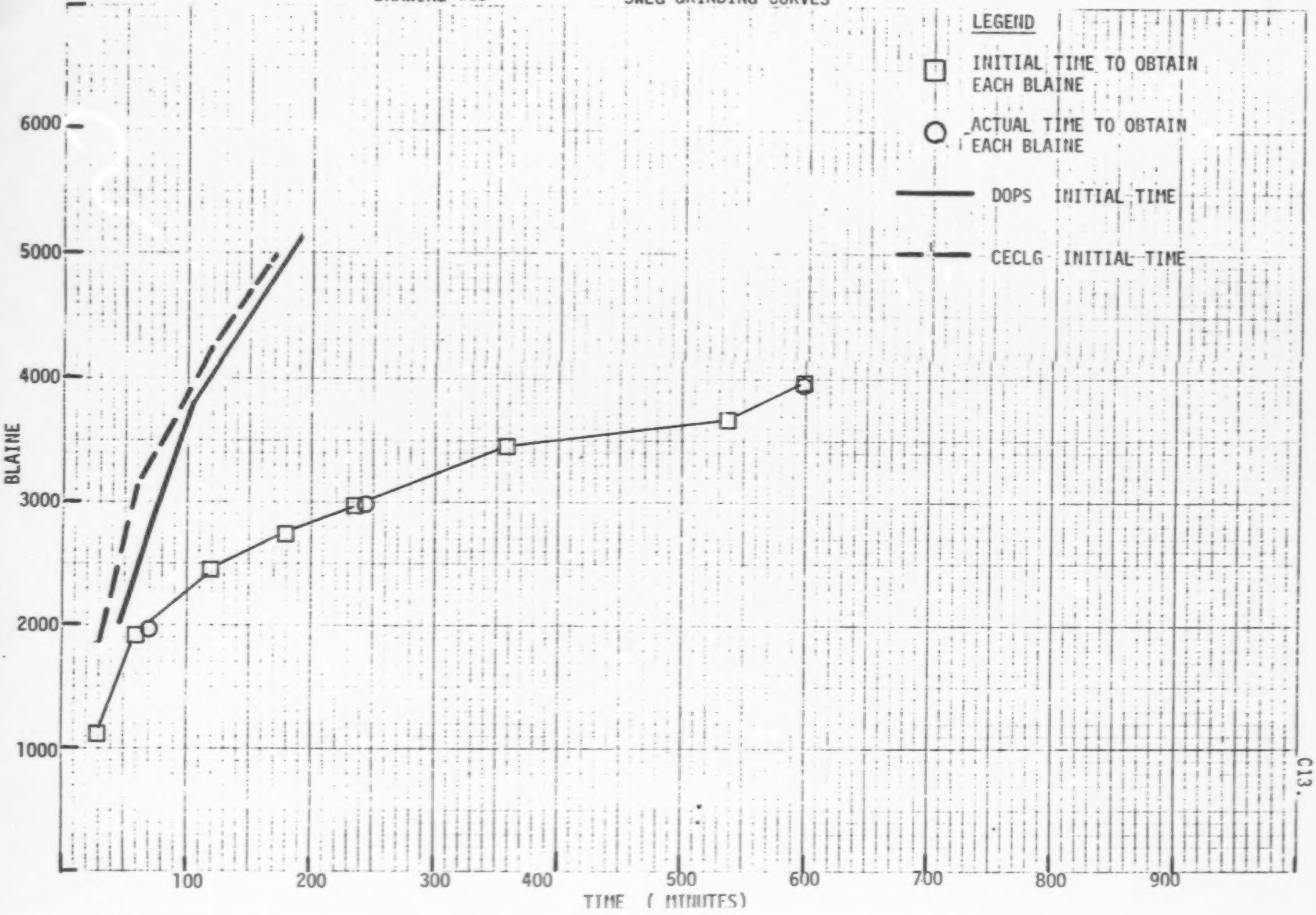
SWLG GRINDING CURVES



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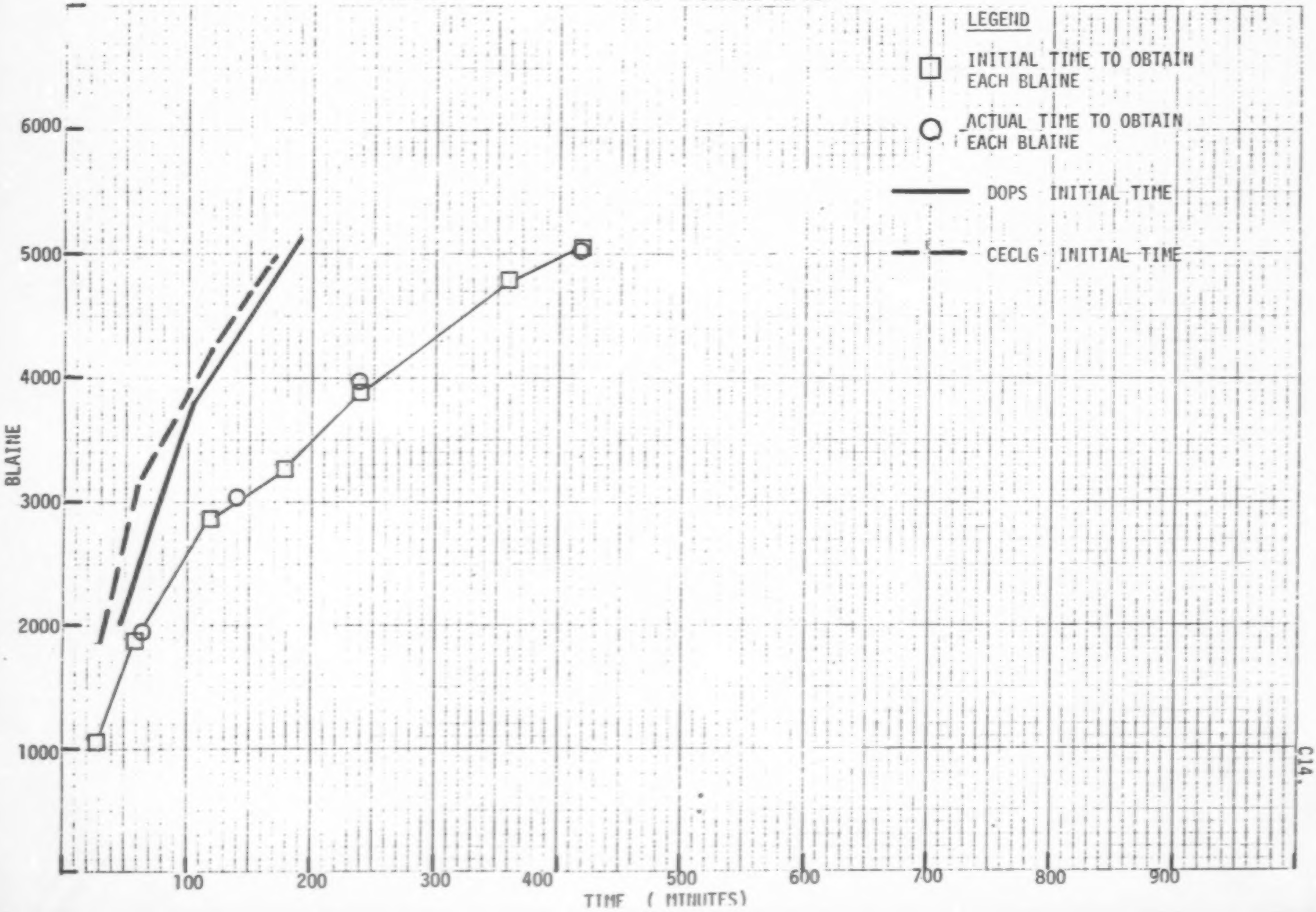

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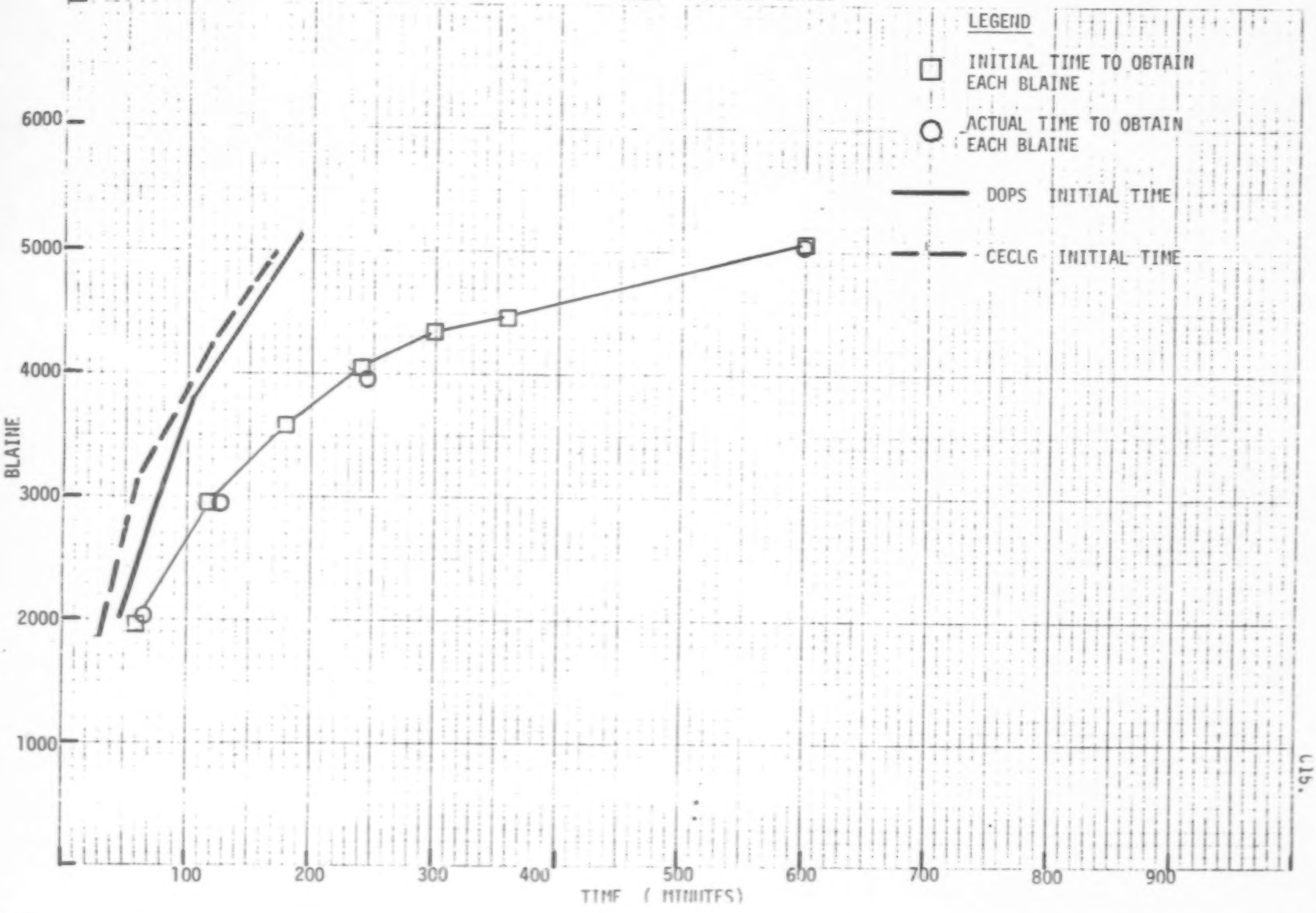

ACKNOWLEDGEMENTS

The co-operation of the non-ferrous metal producers involved, the Scientific Authority (E. Douglas), and CANMET Construction Materials Section (V.M. Malhotra, Head), throughout Phase I of the study is gratefully acknowledged. Technical Service Laboratories, Standard Slag Cement, Building Research Establishment, and Ontario Research Foundation supplied important technical assistance to Trow Ltd. in completing Phase I of the study.

TROW LTD.
John J. Emery

Trow Ltd

FINAL REPORT

PHASE II
CHARACTERIZATION OF NON-FERROUS SLAGS FROM CANADIAN SOURCES
26 SQ. 23440-2-9095

PREPARED FOR

MINERAL SCIENCES LABORATORIES

CANADA CENTRE FOR MINERAL AND ENERGY TECHNOLOGY

JOHN J. EMERY

TROW LTD

MICROMEDIA

44 pp

PROJECT: T3736-ATS
NOVEMBER 19, 1984

43 BAYWOOD ROAD
REXDALE, ONTARIO
M9V 3Y8
(416) 749-1290

SUMMARY

PHASE II
CHARACTERIZATION OF NON-FERROUS SLAGS FROM CANADIAN SOURCES

This report gives the findings of Phase II of the study on the characterization of non-ferrous slags from Canadian sources for cementitious applications. The overall study is concerned with the technical aspects of using ground non-ferrous slags as a partial replacement for portland cement in concrete and cemented mine backfill, which was the focus of Phase II. A total of eighteen concrete mixes incorporating various ground non-ferrous slags (50 percent absolute volume replacement of portland cement) were prepared and tested for compressive strength development at 7, 28 and 90 days with comparisons to a conventional concrete mix. Two cemented mine backfills (cemented tailings and cemented rockfill) incorporating ground non-ferrous slags as a partial replacement for portland cement were also tested.

The test results for concrete mixes incorporating a range of ground nickel, copper and lead slags to replace portland cement (50 percent absolute volume replacement) indicate some promise for mass concrete application where long curing times are acceptable. This should be considered further for lower portland cement replacement levels, including durability and economic considerations. The information developed for similar applications with cemented mine backfills (tailings and rockfill) must be considered preliminary and not of great promise. If the economics are favourable in terms of competing cementitious materials, then further developmental work should be encouraged by mines with non-ferrous slag sources.

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INTRODUCTION

Evaluating the technical feasibility of the partial replacement of portland cement in concrete mixes and cemented mine backfill with typical ground Canadian nickel, copper and lead slags (with two foreign sources for comparison purposes) is the overall objective of a two phase study at Trow Ltd that complements research in the CANMET Construction Materials Section. The previously completed and reported Phase I [1,2] was concerned with: developing a sampling program for non-ferrous slags; visiting Canadian nickel, copper and lead smelters to organize slag sampling and to become familiar with smelter processes; grinding the non-ferrous slags to Blaine fineness levels of 2000, 3000, 4000 and 5000, with comparison to cement clinker and pelletized iron blast furnace slag in terms of grinding energy requirements; chemical and compositional characterization of the non-ferrous slags; and pozzolonic and hydraulic index testing of ASTM C109 mortar cubes (mainly ASTM C595 and C989 procedures) incorporating the ground non-ferrous slags at various portland cement replacement levels. The Phase II study reported herein follows from the Phase I findings, and involved the testing of eighteen typical concrete mixes incorporating ground non-ferrous slags at 50 percent absolute volume replacement of portland cement and evaluation of two cemented mine backfills (cemented tailings and cemented rockfill) incorporating ground non-ferrous slags as a partial replacement for portland cement.

As the Phase II Study was based largely on Phase I, the Phase I findings should be summarized before describing the Phase II program. It appears that the major factor contributing to decreased non-ferrous slag grindability is increasing degree of vitrification. Optical microscopy and qualitative X-ray diffraction were used to determine degree of vitrification (glass contents) without success, except for vitrified iron blast furnace slag. For this reason, a better research method such as the CANMET image analysis method should be adopted to determine the degree of vitrification of non-ferrous slags. The ASTM C989 index (50 percent portland cement replacement and 28 days curing at 23°C) for 4000 Blaine was found to be the best indicator of potential cementitious behaviour. Strength development, as monitored for ASTM C989 curing conditions (essentially ambient), generally increases with increasing vitrification. The strength development results were considered promising and it was recommended that the Phase II study on concrete mixes and tailing mixes incorporating ground non-ferrous slags be completed. A CANMET Seminar was held on May 10, 1984 in Sudbury to cover the Construction Materials Section's overall non-ferrous slag work [2].

Based on the Phase I study results on the cementitious properties of non-ferrous slags from Canadian sources (two foreign non-ferrous slags included for comparison), the following Phase II study program was developed with the Scientific Authority:

(a) Eleven concrete mixes incorporating ground non-ferrous slag to be tested for compressive strength development at 7, 28 and 90 days. The non-ferrous slags to be used are INCO nickel slag, ambient cooled (INCA 3000 and INCA 4000 - see glossary in Table 1); Noranda copper slag, ambient cooled (NRCA 3000 and NRCA 4000); Noranda copper slag, remelted and quenched (NRCR 3000 and NRCR 4000); Kidd Creek copper slag, granulated (KCCG 3000 and KCCG 4000); Belledune lead slag, granulated (BDLG 3000 and BDLG 4000); and Mount Isa copper slag, granulated (MICG 4000).

(b) Two tailings mixes for cemented fill with 70 percent solids by weight to be tested for compressive strength. The mixes are to be made with: 1) Kidd Creek tailings and granulated Kidd Creek copper slag ground to 3000 Blaine (KCCG 3000); and 2) Noranda tailings (NRT-Appendix A) with remelted and quenched Noranda copper slag ground to 4000 Blaine (NRCR 4000).

(c) The standard concrete mix design to be adopted with 50 percent absolute volume replacement of portland cement by ground non-ferrous slag is:

strength	30 MPa at 28 days
portland cement	285 kg
water	140 kg
sand	835 kg
20 mm stone	1150 kg
WRA	1260 ml
air	-
slump	20-80 mm
W/C	0.49

TABLE 1

GLOSSARY FOR SAMPLES INVOLVED IN PHASE II

GROUND NON-FERROUS SLAGS

- BDLG - Belledune, N.B., lead blast furnace slag, fresh, granulated. Blaines of 3000 and 4000. (Brunswick Mining and Smelting Corporation)
- ICNA - Sudbury, Ont., nickel reverberatory slag, fresh, ambient cooled. Blaines of 3000 and 4000. (INCO Metals Company)
- ICNS - Sudbury, Ont., nickel slag (similar to ICNA), ground to cement fineness and stored by INCO. (INCO Metals Company)
- KCCG - Timmins, Ont., copper smelter slag, fresh, granulated. Blaines of 3000 and 4000. (Kidd Creek Mines Ltd.)
- KCCR - Timmins, Ont., copper smelter slag (KCCG) + 20% CaO remelted in laboratory, vitrified by quenching. Blaine of 3000. (Kidd Creek Mines Ltd.)
- NMCA - Murdochville, Qué., copper reverberatory slag, fresh, ambient cooled. Blaine of 4000. (Mines Noranda Limitée)
- NRCA - Noranda, Qué., copper reverberatory slag, fresh, ambient cooled. Blaines of 3000 and 4000. (Noranda Mines Limited)
- NRCR - Noranda, Qué., copper reverberatory slag (NRCA) remelted in laboratory, vitrified by quenching. Blaines of 3000 and 4000. (Noranda Mines Limited)
- MICG - Mount Isa, Australia, copper reverberatory slag, fresh, granulated. Blaine of 4000. (Mount Isa Mines Limited)

CEMENT

- TYPE 10-Portland cement meeting requirements of CSA Type 10. (Canada Cement Lafarge)

ROCKFILL AND TAILINGS

- KCRF - Timmins, Ont., rockfill aggregate as described in Appendix B. (Kidd Creek Mines Ltd.)
- NRT - Noranda, Qué., slag flotation tailings as described in Appendix A. (Noranda Mines Limited)

AGGREGATES

- FINE AGGREGATE - Sand meeting the CSA A23.1 requirements for fine aggregate.
- COARSE AGGREGATE - Minus 19 mm limestone meeting the CSA A23.1 requirements for 19 mm to 4.75 mm size coarse aggregate.

Subsequently, as requested by the Scientific Authority, Kidd Creek copper slag, plus 20 percent CaO remelted in the laboratory, and vitrified by quenching, at 3000 Blaine was added (KCCR 3000). During preliminary mix designs,and to examine the influence of water reducing admixtures (WRA),a further six concrete mixes were added by Trow Ltd to give a total of eighteen plus three control mixes. Further, it was agreed with the Scientific Authority that the Kidd Creek tailings mix should be replaced by a Kidd Creek consolidated rockfill (KCRF) mix as this is their current practice as summarized in Appendix B.

As a detailed literature review was given in Phase I, it will not be repeated here except to note that there is significant interest developing in cemented mine backfill [3] with a number of excellent conference proceedings [4,5] and papers [6] available. Several references indicate an interest in the use of blended cements such as portland cement/fly ash, portland cement/ground non-ferrous slag (current practice at Mount Isa), portland cement/iron blast furnace slag cement,and the use of various activators.

SUMMARY OF PHASE I CHARACTERIZATION AND MORTAR CUBE TESTING RESULTS

For background, the Phase I characterization and testing results for the ground non-ferrous slags involved in Phase II (given in Table 1 glossary) are summarized in Tables 2 through 7 as follows:

Table 2 - Chemical Analysis of Samples
Table 3 - XRD Analysis of Samples
Table 4 - Grinding Data for Samples
Table 5 - "C109" Mortar Cube Compressive Strength Development at 23°C 50 Percent Absolute Volume Replacement of Portland Cement (ASTM C989)
Table 6 - "C109" Mortar Cube Compressive Strength Development, 28 Days curing at 35°C 20, 35, 50 and 70 Percent Absolute Volume Replacement of Portland Cement (ASTM C595)
Table 7 - Summary of Results from Phase I for Samples (4000 Blaine) Involved in Phase II

The data in Table 7 are particularly important as they indicate a potential for cementitious use at the 50 percent portland cement replacement level (absolute volume replacement throughout for concrete mixes) for MICG 4000, KCCG 3000,

TABLE 2

CHEMICAL ANALYSIS OF SAMPLES

PERCENT

	BDLG	ICNA	KCCG	KCCR	NMCA	NRCA	NRCR	MICG
SiO_2	19.91	35.66	26.54	22.10	36.26	34.51	36.78	42.76
Al_2O_3	1.66	5.59	3.74	2.94	2.16	6.55	7.16	3.35
Fe_2O_3	45.35	52.99	60.07	44.36	49.00	49.50	50.00	45.79
CaO	16.60	2.74	2.09	20.32	7.30	2.20	1.93	8.16
MgO	0.68	2.53	1.63	1.52	1.91	1.48	1.49	2.92
Na_2O	0.26	0.87	0.33	0.23	0.28	1.02	1.06	0.21
K_2O	0.25	0.65	0.33	0.98	1.26	1.01	0.96	0.65
TiO_2	0.14	0.32	0.14	0.12	0.13	0.35	0.39	0.24
MnO	0.09	0.07	0.10	0.08	0.08	0.10	0.09	0.05
Cu			1.10		0.48	0.43	0.41	
Ni		0.23						
Pb	4.84							0.06
LOI	-5.75	-5.32	-5.84	-4.39	-3.91	-5.24	-6.13	-3.44
S	1.30	1.46	1.28	0.45	1.01	1.20	1.13	0.56

NOTES:

1. Major oxides by whole rock analyses - inductively coupled argon plasma direct reading spectrometer system (TSL).
2. P_2O_5 not determined due to severe spectral interference from Cu and Zn.
3. Total iron expressed as Fe_2O_3, negative LOI indicates FeO present.
4. LOI at 700°C.
5. S by LECO.
6. Average value for two determinations generally given.

TABLE 3

XRD ANALYSIS OF SAMPLES

SAMPLE	RESULTS
BDLG	Glassy slag. The crystalline material comprises a melilite-type of mineral and wustite (FeO).
INCA	Contains abundant crystalline material. The main minerals are fayalite (Fe-rich olivine) and magnetite (Fe_3O_4).
KCCG	Very glassy slag. The small amount of crystalline material comprises olivine (probably fayalite) and magnetite.
KCCR	Contains more crystalline material than the fresh, granulated slag. The crystalline material comprises fayalite.
NMCA	Contains abundant crystalline material. The main minerals present are olivine, diopside and magnetite.
NRCA	Contains abundant crystalline material. The main minerals present are fayalite and magnetite.
NRCR	Very glassy slag. The small amount of crystalline material is fayalite.
MICG	Contains abundant crystalline material. The main mineral present is quartz; also contains a few percent fayalite.

NOTE: Siemens X-Ray equipment used for general analysis for main minerals (BRE).

TABLE 4

GRINDING DATA FOR SAMPLES

	BDLG	ICNA	KCCG	KCCR	NMCA	NRCA	NRCR	MICG
BLAINE	BLAINE AND TIME IN MINUTES							
3000	2970 180 (72)	3044 75(303)	2976 195 (78)	3168 140 (56)	3104 120 (48)	3107 118 (47)	3150 122 (49)	2981 105 (42)
4000	3981 653(261)	4047 115 (46)	4014 600(240)	4095 210 (84)	3955 190 (76)	3927 190 (76)	4138 240 (96)	4118 240 (96)
SG	4.00	3.54	3.90	3.68	3.73	3.53	3.39	3.40
% PASSING 45 µm								
3000	85.55	83.68	79.48	-	82.80	77.36	90.41	87.97
4000	87.59	91.49	80.92	91.36	87.50	82.61	92.23	93.88

NOTE:

1. Average value for two Blaine and time determinations generally given.
2. The computed average energy consumption (kWh/t) for production grinding is given in brackets. This is based on a laboratory mill to typical production mill scale-up of 1 minute=0.4 kWh/t for the mills used, and F.L. Smidth data. (Nielsen, H.C. Alsted, Utilization of Fly Ash and Slag in the Cement Industry, Poster, 7th International Congress on the Chemistry of Cement, Paris, 1980.)
3. Two ground samples blended after Blaine values determined, then % passing 45 µm determined.
4. SG determined for 4000 Blaine, using air-comparison pycometer.
5. Nominal Blaine fineness of slags indicated - 3000 and 4000.

TABLE 5

"C109" MORTAR CUBE COMPRESSIVE STRENGTH DEVELOPMENT AT 23°C

50 PERCENT ABSOLUTE VOLUME REPLACEMENT OF PORTLAND CEMENT

(ASTM C989)

MPa FLOW

	BDLG	ICNA	KCCG	KCCR	NMCA	NRCA	NRCR	MICG
CONTROLS 7D	31.3 111	31.3 111	31.3 111	31.3 111	31.3 111	31.3 111	31.3 111	31.3 111
28D	36.8	36.8	36.8	36.8	36.8	36.8	36.8	36.8
3000 Blaine								
Days 1	8.5 115	4.0 109	4.9 112	5.0 110	4.7 115	3.3 113	5.0 106	4.0 109
7	12.4	8.8	10.3	11.0	10.9	9.5	12.6	8.8
28	23.5	15.6	25.7	21.8	15.7	18.0	24.6	19.9
90	37.9	26.0	29.7	26.5	20.6	25.9	32.0	24.9
4000 Blaine								
Days 1	12.1 113	4.1 113	5.6 107	6.3 115	5.0 113	4.3 107	5.6 107	4.2 105
7	14.1	10.0	10.5	12.5	12.0	11.0	14.8	12.2
28	29.8	17.5	26.7	24.8	19.2	20.5	30.5	26.7
90	40.2	28.3	31.4	31.1	23.2	29.2	37.3	34.8
INDEX 7 Day/28 Day								
3000 Blaine	0.40/0.64	0.28/0.42	0.33/0.70	0.35/0.59	0.35/0.43	0.30/0.49	0.40/0.67	0.28/0.54
4000 Blaine	0.45/0.81	0.32/0.48	0.34/0.73	0.40/0.67	0.38/0.52	0.35/0.56	0.47/0.83	0.39/0.73

NOTES:

1. Portland cement meets requirements of ASTM C989.
2. Nominal Blaine fineness of slags indicated - 3000 and 4000.
3. All compressive strengths in MPa, with flow indicated for each mix.
4. INDEX is the 7 day and 28 day ratio of slag/cement mortar cube compressive strength to cement mortar cube (control) compressive strength.

TABLE 6

"C109" MORTAR CUBE COMPRESSIVE STRENGTH DEVELOPMENT, 28 DAYS CURING AT 38°C

20, 35, 50 AND 70 PERCENT ABSOLUTE REPLACEMENT OF PORTLAND CEMENT

(ASTM C595)

MPa FLOW

	BDLG	ICNA	KCCG	KCCR	NMCA	NRCA	NRCR	NRCG	MICG
CONTROL	29.6 111	29.6 111	29.6 111	29.6 111	29.6 111	29.6 111	29.6 111	29.6 111	29.6 111
20% Replacement									
3000 Blaine	22.7 114	24.0 106	23.4 112	23.6 114	22.1 115	25.2 114	23.9 110		22.5 113
4000 Blaine	30.5 114	24.8 112	25.4 114	25.8 114	24.6 115	26.5 115	25.7 105		24.5 106
35% Replacement									
3000 Blaine	16.1 106	21.5 112	20.4 114	21.3 113	19.8 112	23.7 105	21.5 113		17.5 111
4000 Blaine	28.5 107	24.5 106	20.3 112	22.5 107	20.8 111	25.3 114	23.8 105		20.6 113
50% Replacement									
3000 Blaine	15.0 114	19.1 115	15.5 114	18.0 113	18.0 108	22.7 105	20.2 114		14.0 109
4000 Blaine	20.6 108	21.4 105	17.8 105	18.8 107	19.7 110	23.9 105	23.6 108		15.5 109
70% Replacement									
3000 Blaine	7.9 115	12.8 115	8.5 107	12.8 106	10.5 110	14.5 115	12.5 109		6.8 114
4000 Blaine	8.5 108	17.1 113	9.1 111	13.4 105	13.9 106	17.5 111	16.0 105		9.4 115
PI/HI									
3000 Blaine	0.54/0.27	0.73/0.43	0.69/0.29	0.72/0.43	0.67/0.35	0.80/0.49	0.73/0.42		0.59/0.23
4000 Blaine	0.96/0.29	0.83/0.58	0.69/0.31	0.76/0.45	0.70/0.47	0.85/0.59	0.80/0.54		0.70/0.32

NOTES:

1. Canada Cement Lafarge Type 10 portland cement.
2. Nominal Blaine fineness of slags indicated - 3000 and 4000.
3. All compressive strengths in MPa, with flow indicated for each mix.
4. PI and HI are the pozzolanic activity index and slag activity index with cement, respectively, in accordance with ASTM C595.

TABLE 7

SUMMARY OF RESULTS FROM PHASE I FOR SAMPLES (4000 BLAINE) INVOLVED IN PHASE II

	SLAG	C989 28 DAY INDEX	C595 POZZOLANIC ACTIVITY INDEX	C595 SLAG ACTIVITY INDEX	C989 90 DAY STRENGTH MPa	Vitrification
1.	ICNA	0.48	0.83	0.58	28.3	crystalline
2.	NMCA	0.52	0.70	0.47	23.2	crystalline
3.	NRCA	0.56	0.85	0.59	29.2	crystalline
4.	KCCR	0.67	0.76	0.45	31.1	glassy/crystalline
	(3000 Blaine	0.59)			(26.5)	
5.	MICG	0.73	0.70	0.32	34.8	glassy
6.	KCCG	0.73	0.69	0.31	31.4	very glassy
	(3000 Blaine	0.70)			(29.7)	
7.	BDLG	0.81	0.96	0.29	40.2	very glassy
	(3000 Blaine	0.64)			(37.9)	
8.	NRCR	0.83	0.80	0.54	37.3	very glassy
	(3000 Blaine	0.67)			(32.0)	

NOTES:

1. Rank is in order of increasing C989 28 day index from Table 5.
2. C595 pozzolanic activity index and C595 slag activity index for Table 6.
3. C989 90 day strength from Table 5.
4. Vitrification from Table 3.

KCCG 4000, BDLG 4000 and NRCR 4000, using a C989 index ≥ 0.70 as the criteria, noting that this is current practice at Mount Isa Mine (MICG) which did not have the highest C989 index. Strength development generally increases with increasing vitrification, but increased grinding energy/costs will also be involved. A balance between desired strength development and grinding economics is thus required. For this reason, lower portland cement replacement levels and/or Blaine for ground non-ferrous slags may be desirable in practice. Lower replacement levels may also permit the use of less glassy, ground non-ferrous slags such as ICNA and NRCA in cementitious applications.

CONCRETE MIXES INCORPORATING GROUND NON-FERROUS SLAG

Full details on the concrete mixes incorporating ground non-ferrous slags and their compressive strength development are given in Tables 8, 9 and 10. All of the concrete mixes were prepared and tested in accordance with CSA CAN3-A23.1 M77 (Concrete Materials and Methods of Concrete Construction) and CSA CAN3-A23.2 M77 (Methods of Test for Concrete). The gradations of the coarse and fine aggregates for the concrete mixes meet CSA A23.1 requirements as shown in Table 8.

Mix details and compressive strengths for the twelve primary mixes and three control mixes are given in Table 9. Control 3 (repeat of Control 2) was used for 'comparison' purposes throughout as 100 mm ϕ by 200 mm cylinders were involved and it gives compressive strengths in the range anticipated for the Control 1 150 mm ϕ by 300 mm cylinders [7]. In this regard, the Control 2 compressive strengths are considered somewhat high which required an additional testing period to obtain Control 3 results. Mix details and compressive strengths for the six supplemental mixes (additional non-ferrous slags and some of the primary mixes with water reducing admixture) are given in Table 10.

From the data given in Tables 9 and 10, it is possible to compare all of the concrete mixes incorporating ground non-ferrous slags in terms of both compressive strength development (7, 28 and 90 days) and compressive strength ratio to the control (7, 28 and 90 days). This has been done in Table 11 for compressive strength development and ratio, and in Figures 1 and 2 for compressive strength development and compressive strength ratio, respectively.

TABLE 8

GRADATIONS OF COARSE AND FINE AGGREGATES FOR CONCRETE MIXES

	SIEVE SIZE	PERCENT PASSING	CSA A23.1 REQUIREMENTS PERCENT PASSING
COARSE AGGREGATE	25.4 mm	100	100
	19 mm	95	90-100
	12.7 mm	52	30-75
	9.5 mm	36	20-55
	4.75 mm	2	0-10
FINE AGGREGATE	9.5 mm	100	100
	4.75 mm	97	95-100
	2.36 mm	85	80-100
	1.18 mm	72	50-85
	600 μm	52	25-60
	300 μm	22	10-30
	150 μm	6	2-10

TABLE 9

CONCRETE MIX DETAILS AND COMPRESSIVE STRENGTH DEVELOPMENT

PRIMARY MIXES AND CONTROLS

Ground Non-Ferrous Slags Incorporated as Fifty Percent Absolute Volume Replacement of Type 10 Portland Cement

MIX	BDLG 3000	BDLG 4000	INCA 3000	INCA 4000	KCCG 3000	KCCG 4000	KCCR 3000	NRCA 3000	NRCA 4000	NRCR 3000	NRCR 4000	MICG 4000 (Foreign)	CONTROL 1	CONTROL 2	CONTROL 3 (Used as Control)
MIX DETAILS															
Ground Non-Ferrous Slag, kg	8.32	8.32	7.36	7.36	8.09	8.09	7.04	7.31	7.31	4.77	4.77	6.54	-	-	-
Type 10 Cement, kg	6.54	6.54	6.54	6.54	6.54	6.54	6.04	6.54	6.54	4.45	4.45	6.04	13.09	13.09	13.09
Fine Aggregate, kg	38.41	38.41	38.41	38.41	38.41	38.41	35.54	38.41	38.41	26.05	26.05	35.54	38.41	38.41	38.41
Coarse Aggregate,kg	52.86	52.86	52.86	52.86	52.86	52.86	48.96	52.86	52.86	35.91	35.91	48.96	52.86	52.86	52.86
Water, kg	7.28	7.28	6.82	6.82	7.17	7.17	6.41	6.80	6.80	4.50	4.50	6.18	6.46	6.46	6.46
WRA, ml	-	-	-	-	-	-	-	-	-	-	-	-	60.7 (Mulco A)	60.7 (Mulco A)	60.7 (Mulco A)
Slump, mm	130	130	75	75	110	120	50	70	100	55	50	65	45	40	70.0
Air, %	1.5	1.5	2.2	1.9	1.5	1.9	1.7	1.5	1.4	1.9	1.9	1.9	3.9	3.2	3.4
Temperature, °C	20	19	20	20	20	20	20	20	20	20	21	20	20	21	19
Bulk Density, kg/m^3	1937	1963	1949	1923	1958	1953	1937	1958	1968	1930	1940	1928	1928	1977	1958
COMPRESSIVE STRENGTHS, MPa															
7 Day	9.5	12.4	9.2	10.7	9.9	10.2	10.7	9.7	9.2	10.7	13.8	11.0	27.6	32.0	29.2
	9.3	11.9	9.2	10.2	9.4	11.1	10.5	9.6	9.2	11.0	14.0	11.2	27.3	31.0	29.8
	10.0	12.1	9.7	10.7	9.5	10.4	10.7	9.4	9.5	11.0	13.8	11.5	28.1	32.0	28.3
Avg.	9.6	12.1	9.4	10.5	9.6	10.6	10.6	9.6	9.3	10.9	13.9	11.2	27.7	31.6	29.1
28 Day	23.7	28.5	16.8	20.1	22.4	24.7	21.4	17.3	18.4	23.4	27.5	24.5	35.6	40.8	37.4
	21.4	27.3	17.3	20.4	22.4	24.7	21.7	19.4	18.9	23.2	27.0	26.0	35.9	41.0	37.9
	21.7	30.3	16.8	20.9	21.9	26.0	21.2	18.4	18.9	24.0	26.3	24.5	-	40.5	38.1
Avg.	22.2	28.7	17.0	20.5	22.2	25.1	21.4	18.4	18.6	23.5	26.9	25.0	35.8	40.8	37.8
90 Day	32.6	40.8	23.4	28.5	28.0	28.5	31.6	24.5	24.0	27.5	31.1	30.6	42.2	49.0	45.4
	35.6	39.7	24.0	28.0	28.0	30.1	30.1	25.5	25.0	27.5	31.2	30.1	42.3	47.3	44.2
	35.7	42.8	22.9	27.0	27.5	29.6	31.1	24.5	26.0	27.0	31.1	31.1	-	47.7	44.5
Avg.	34.6	41.1	23.4	27.8	27.8	29.4	30.9	24.8	25.0	27.3	31.1	30.6	42.2	48.0	44.7

NOTE: All mixes and tests in accordance with CSA A23.1 and CSA A23.2, respectively. 100 mm ɸ by 200 mm cylinders for all mixes except CONTROL 1 (150 mm ɸ by 300 mm cylinders).

TABLE 10

CONCRETE MIX DETAILS AND COMPRESSIVE STRENGTH DEVELOPMENT

SUPPLEMENTAL MIXES

Ground Non-Ferrous Slags Incorporated as Fifty Percent Absolute Volume Replacement of Type 10 Portland Cement

MIX	INCA 3000 WRA	INCA 4000 WRA	INCS	INCS WRA	NMCA 4000 WRA	NRCA 3000 WRA
MIX DETAILS						
Ground Non-Ferrous Slag, kg	7.36	7.36	7.14	7.14	7.77	7.32
Type 10 Cement, kg	6.54	6.54	6.54	6.54	6.54	6.54
Fine Aggregate, kg	38.41	38.41	38.41	38.41	38.41	38.41
Coarse Aggregate, kg	52.86	52.86	52.86	52.86	52.86	52.86
Water, kg	6.46	6.46	6.46	6.46	6.46	6.46
WRA, ml	64.5 (TCDA A)	64.5 (TCDA A)	-	63.5 (Mulco A)	66.4 (Mulco A)	64.3 (TCDA A)
Slump, mm	110	90	40	90	90	90
Air, %	4.0	3.3	2.5	5.3	3.3	3.7
Temperature, °C	21	21	20	22	21	21
Bulk Density, kg/m^3	1967	1936	1954	1913	1937	1967
COMPRESSIVE STRENGTHS, MPa						
7 Day	12.2	12.0	14.8	14.8	15.0	11.2
	12.2	11.7	13.8	14.8	14.5	11.7
	11.7	12.4	13.8	14.0	14.8	11.7
Avg.	12.0	12.0	14.1	14.5	14.8	11.5
28 Day	21.4	24.5	22.9	24.0	23.4	22.9
	21.9	24.5	22.4	23.2	24.0	24.0
	21.4	25.0	23.4	22.9	23.4	24.0
Avg.	21.6	24.7	22.9	23.9	23.6	23.6
90 Day	30.6	31.1	31.1	30.1	27.8	30.6
	29.6	31.6	30.1	31.1	28.8	31.6
	29.1	31.1	30.1	31.2	28.3	31.7
Avg.	29.8	31.3	30.4	30.8	28.3	31.3

NOTE: All mixes and tests in accordance with CSA A23.1 and CSA A23.2, respectively, 100 mm ϕ by 200 mm cylinders.

TABLE 11

CONCRETE MIX COMPRESSIVE STRENGTH SUMMARY

Ground Non-Ferrous Slags Incorporated as Fifty Percent Absolute Volume Replacement of Type 10 Portland Cement

MIX	SLUMP mm	AIR %	COMPRESSIVE STRENGTH AND RATIO TO CONTROL 3 7 DAY AVG. MPa	 7 DAY AVG. Ratio	 28 DAY AVG. MPa	 28 DAY AVG. Ratio	 90 DAY AVG. MPa	 90 DAY AVG. Ratio
BDLG 3000	130	1.5	9.6	0.33	22.2	0.59	34.6	0.77
BDLG 4000	130	1.5	12.1	0.42	28.7	0.76	41.1	0.92
INCA 3000	75	2.2	9.4	0.32	17.0	0.45	23.4	0.52
INCA 4000	75	1.9	10.5	0.36	20.5	0.54	27.8	0.62
KCCG 3000	110	1.5	9.6	0.33	22.2	0.59	27.8	0.62
KCCG 4000	120	1.9	10.6	0.36	25.1	0.66	29.4	0.66
KCCR 3000	50	1.7	10.6	0.36	21.4	0.57	30.9	0.69
NRCA 3000	70	1.5	9.6	0.33	18.4	0.49	24.8	0.55
NRCA 4000	100	1.4	9.3	0.32	18.6	0.49	25.0	0.56
NRCR 3000	50	1.9	13.9	0.48	26.9	0.71	31.1	0.70
NRCR 4000	55	1.9	10.9	0.37	23.5	0.62	27.3	0.61
MICG 4000	65	1.9	11.2	0.38	25.0	0.66	30.6	0.68
			(Avg. 0.36)			(Avg. 0.59)		(Avg. 0.66)
CONTROL 1	45	3.9	27.7	-	35.8	-	42.2	-
CONTROL 2	40	3.2	31.6	-	40.8	-	48.0	-
CONTROL 3	70	3.4	29.1	-	37.8	-	44.7	-
INCA 3000 WRA	110	4.0	12.2	0.42	21.6	0.57	29.8	0.67
INCA 4000 WRA	90	3.3	12.0	0.41	24.7	0.65	31.3	0.70
INCS	40	2.5	14.1	0.48	22.9	0.60	30.4	0.68
INCS WRA	90	5.3	14.5	0.50	23.4	0.62	30.8	0.69
NMCA 4000 WRA	90	3.3	14.8	0.51	23.6	0.62	28.3	0.63
NRCA 3000 WRA	90	3.7	11.5	0.40	23.6	0.62	31.3	0.70
			(Avg. 0.45)			(Avg. 0.62)		(Avg. 0.68)

NOTES:

1. Data summarized From Tables 9 and 10.
2. Ratio is the average compressive strength divided by the Control 3 average compressive strengths at the same age.

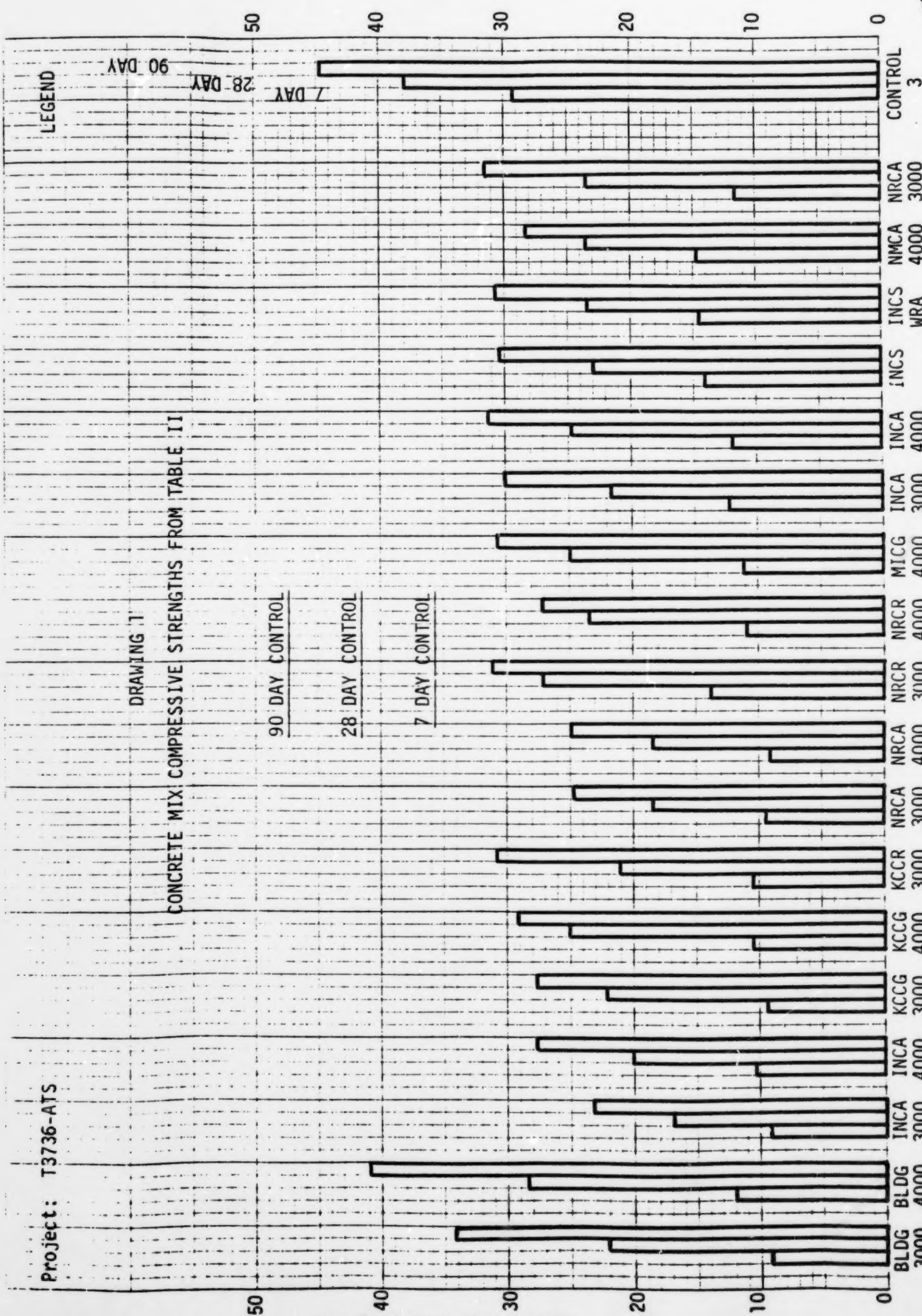
TABLE 12

CEMENTED MINE TAILINGS MIX DETAILS AND COMPRESSIVE STRENGTH DEVELOPMENT

NRT - TAILINGS TYPE 10/NRCR 4000 - CEMENT

MIX DETAILS	4% NRCR 4000/ 4% TYPE 10	8% NRCR 4000/ 4% TYPE 10	12% NRCR 4000/ 4% TYPE 10
NRCR 4000, kg	0.100	0.232	0.300
Type 10 Cement, kg	0.100	0.116	0.100
NRT Tailings, kg	2.300	2.552	2.100
Water, kg	0.750	0.870	0.750
COMPRESSIVE STRENGTHS, MPa			
7 Day			
(Strength not adequate for testing)			
28 Day	2.2	2.3	2.5
	2.1	2.1	2.4
	1.9	2.3	2.5
Avg.	2.1	2.2	2.5
90 Day	1.63	1.62	1.66
	1.66	1.68	1.68
	1.59	1.67	1.69
Avg.	1.63	1.66	1.68

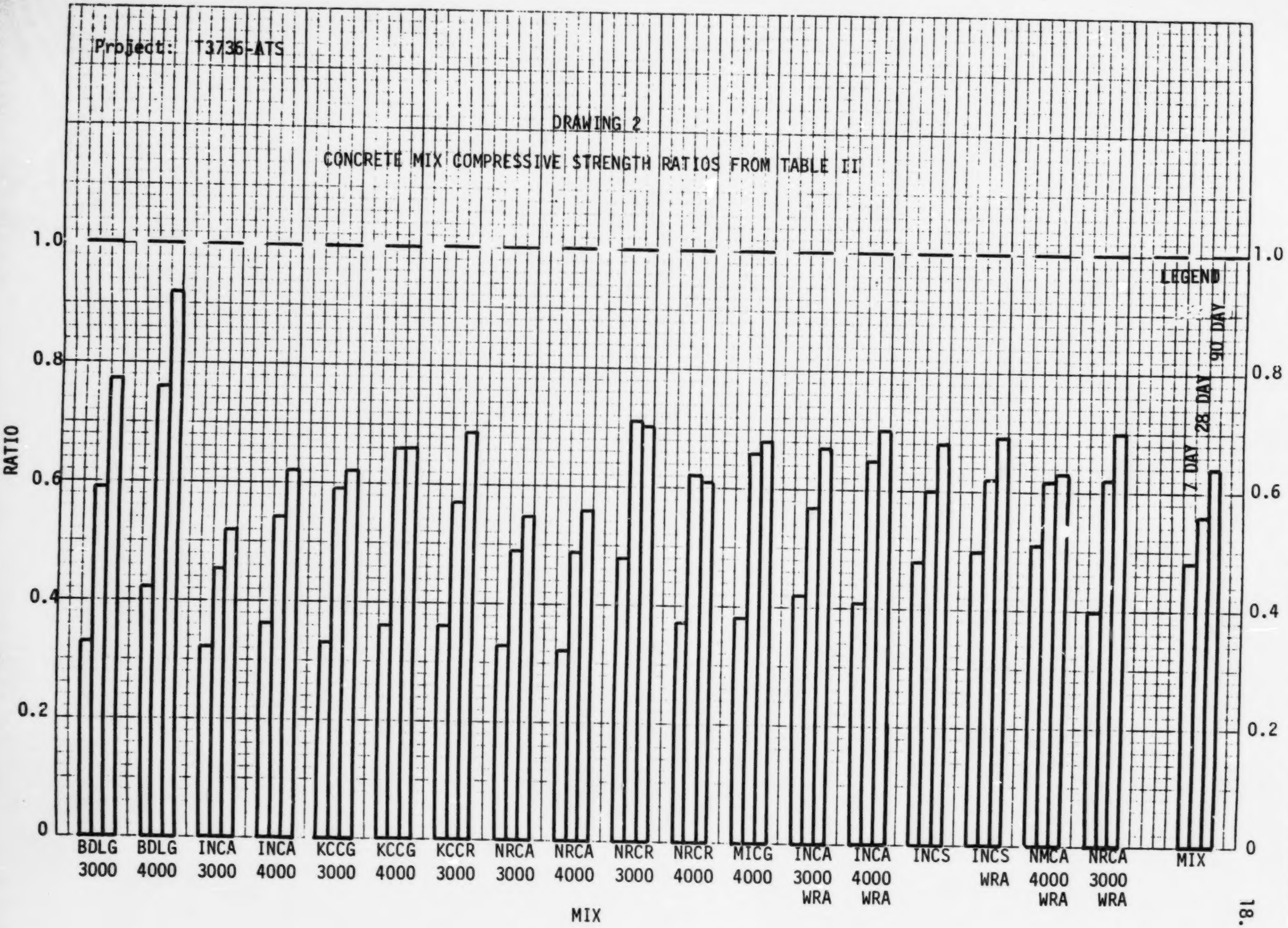
NOTE: Absolute volume replacement not adopted.



Project: T3736-ATS

DRAWING 1

CONCRETE MIX COMPRESSIVE STRENGTHS FROM TABLE II



DRAWING 2

CONCRETE MIX COMPRESSIVE STRENGTH RATIOS FROM TABLE II

While the 7 day compressive strengths of concrete mixes incorporating non-ferrous slags (50 percent absolute volume replacement of Type 10 portland cement) only average 36 percent of the control for the primary mixes (45 percent for supplemented mixes) at 7 days, by 90 days the average is 66% for the primary mixes (68 percent for the supplemental mixes) with BDLG 4000 achieving 92 percent, at 90 days most of the mixes have reached at least 27 MPa (∿ 4000 psi). It would appear that a water reducing admixture helps at early ages but is not significant at later ages. While these results are not that promising for general use of ground non-ferrous slag for portland cement replacement in concrete mixes, they do indicate a potential for such use where adequate time is available for strength development such as mass foundations. Further, lower replacement levels are possible which would give improved strength development. Other factors such as durability and economics must of course also be considered.

CEMENTED MINE TAILINGS AND CEMENTED ROCKFILL

As a limited amount of the ground Noranda non-ferrous slag NRCR 4000 and tailings NRT (Appendix A) were available, 50 mm cubes (uncompacted) at 70 percent solids were adopted for the cemented mine tailings mixes. Signficant bleeding of the cubes in the mold was observed with a cube top shrinkage of about 3 mm in 12 hours. The cubes were demolded as soon as sufficient strength was obtained and stored for curing in the moist room at 23°C. Mix details and compressive strength developed for the cemented mine tailings mixes are summarized in Table 12. It took until 28 days before strengths adequate for proper compressive strength testing were developed. There was little improvement in strength from the 4% NRCR 4000/4% TYPE 10 cementitious system to the 12% NRCR 4000/4% TYPE 10 cementitious system, and a characteristic decrease in strength with increasing curing from 28 days to 90 days. While further testing would be required to confirm these observations, it would appear that the majority of the cementitious action was contributed by the Type 10 portland cement for this cemented mine backfill, and that the lower compressive strength at 90 days may be related to the tailings composition (i.e. chemical reactions during curing) that may not be a factor in actual applications.

For the Kidd Creek cemented rockfill (KCRF) incorporating ground Kidd Creek non-ferrous slag KCCG 3000, the mix was made in accordance with the proportions given in Appendix B in order to simulate the mix actually being used at the mine. The prepared mix with a cementitious system 3% KCCG 3000/2% TYPE 10 (typically used 5% Type 10 portland cement in the past, then a Type 10 portland cement/ground iron blast furnace slag blend more recently and now a Class C fly ash, at the mine) segregated easily and had the appearance of a no-fines concrete. Strength development was slow so that the 150 mm ϕ by 300 cylinder were not removed from their molds until the time to testing in the moist room at 23°C had been achieved. Care was required at all stages of testing to obtain reliable results (this is also commented on by Kidd Creek staff in Appendix B). It took until 2 days before strength adequate for capping and handling were achieved.

The cemented rockfill mix details and compressive strength development are summarized in Table 13. It would appear that the 1.73 MPa strength developed at 90 days is significantly less than that for equivalent portland cement or portland cement ground iron blast furnace slag cementitious systems found satisfactory at the mine of about 3 to 4 MPa (Appendix B). It is clear that significant further developmental work would be required to develop this ground non-ferrous slag application, including an economic assessment.

COMMENTS

The test results for concrete mixes incorporating ground non-ferrous slags to replace portland cement (50 percent absolute volume replacement) indicate some promise for mass concrete applications such as large foundation elements where long curing times are available. This should be considered further for lower portland cement replacement levels, including durability and economic considerations.

The information developed on the use of ground non-ferrous slags in cemented mine fill (tailings and rockfill) must be considered preliminary. If the economics are favourable in terms of competing cementitious materials (fly ash and ground iron blast furnace slag for instance), then further developmental work should be encouraged by mines with suitable non-ferrous slags.

TABLE 13

CEMENTED ROCKFILL MIX DETAILS AND COMPRESSIVE STRENGTH DEVELOPMENT

KCRF - ROCKFILL   TYPE 10/KCCG 3000 - CEMENT

3% KCCG 3000/2% TYPE 10

MIX DETAILS	
KCCG 3000, kg	3.27
Type 10 Cement, kg	2.18
Fine Rockfill, kg	27.27
Coarse Rockfill, kg	81.82
Water, kg	5.46
COMPRESSIVE STRENGTHS	
7 Day	
(Strength not adequate for testing)	
28 Day	1.04
	1.14
	0.88
Avg.	1.02
90 Day	1.59
	1.88
	1.72
Avg.	1.73

NOTE: 1. Some places of segregation observed for test cylinders.

2. Absolute volume replacement not adopted.

ACKNOWLEDGEMENTS

The co-operation of the non-ferrous metal producers involved, the Scientific Authority (E. Douglas), and CANMET Construction Materials Section (V.M. Malhotra, Head), throughout the study is gratefully acknowledged.

TROW LTD
John J. Emery

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APPENDIX A

INFORMATION RECEIVED FOR NRT

Slag Flotation Tailings from Mines Noranda Limiteé, Noranda, Québec

Mines Noranda Limitée
Division Horne
C.P. 4000
Noranda, Qué. J9X 5B6

noranda

January 6th, 1983

Dr. J.J. Emery, Manager
Advanced Technical Services
Trow Limited
43 Baywood Road
REXDALE, Ontario
M9V 3Y8

Dear John:

The following slag samples have been shipped, to your attention, in 45 gallon drums.

Sample A: 2, 45 gallon drums of reverberatory slag.
Sample B: 2, 45 gallon drums of granulated reverberatory slag.
Sample C: Noranda Reactor slag tailings - 100 lbs.

A representative sample of each was taken for chemical analysis which follows:

	Fe, %	SiO_2, %	Cu, %
Sample A	34.3	36.0	0.5
Sample B	34.6	35.3	0.46
Sample C	40.0	24.5	0.39

Background information on source and sampling is given below.

Sample A (Reverberatory Slag)

Reverberatory slag is tapped into ladles and brought to the slag dump where it is poured in the liquid state unto a sloping surface. When cool, it is recovered and transported to the Noranda CN yard where it is crushed and stockpiled. The sample shipped to you was taken half-way up the 50' high stockpile.

Approximately 2 feet of slag was removed from the surface before filling the drums using a shovel from a single location. Samples were taken at regular intervals, while filling the drums, to make up the representative sample for chemical analysis.

This slag sample is representative of the current cooling and handling conditions of our reverberatory slag.

Sample B (Granulated Reverberatory Slag)

This sample was taken from a pile (very approximate 75,000 tons) of slag which was granulated some 8 years back. After removing some slag from the surface (top of the pile) the drums were filled using a shovel from a single location. A representative sample was taken for chemical analysis.

Sample C

Slag from the Noranda reactor is slow cooled, milled and treated by a flotation process to remove the copper content and generate a disposable slag tailing. Representative slag tailing samples are sent to the analytical laboratory on an 8 hour basis. These samples were collected from a 1-month period to make up Sample C.

I hope these samples and the corresponding information are sufficient for the study. If not do not hesitate to give me a call.

During your visit at the Horne you briefly described the method of encapsulating dangerous fine materials in concrete. This is an area we are interested and would appreciate receiving information you may have available.

Yours very truly,

D. Poggi
Metallurgical Superintendent

DP/dv

cc L.A. Mills

APPENDIX B

INFORMATION RECEIVED FOR KCRF

Rockfill Aggregates from Kidd Creek Mines Ltd., Kidd Creek, Ontario

Kidd Creek Mines Ltd.
B-2002, Timmins, Ontario P4N 7K1
Mine Site (705) 267-2161

February 7, 1984

Dr. J. Emery, P.Eng.
Trow Ltd.
43 Baywood Road
Rexdale, Ontario
M9V 3Y8

Dr. Emery;

I am forwarding to you several reprints of our most recent paper, entitled "Backfill Practice and Technology at Kidd Creek Mines", which was published in the August 1983 CIM Bulletin.

Mark Adams contacted me and requested that Kidd Creek ship several hundred pounds of sand to your office to allow continued research into the use of ground copper slag as a partial portland cement replacement in backfill. You will remember from our telephone conversation of February 7, 1984, that hydraulic sand fill plays a minor role in the backfill operation at Kidd; last year we placed 50,000 Tonnes of sand fill as opposed to just over 1,000,000 Tonnes of consolidated rock fill. The limited use of sand fill is discussed in the paper which I am sending to you.

We are willing to assist you in this research program, and will send you the required samples of our aggregates, which comprise the bulk of our fill. The aggregate will arrive in four or five, five gallon pails, (approximate weight is 60 pounds each). One pail will contain the -1cm fine aggregate, which should be mixed with the coarse aggregate in a 1:3 ratio by weight.

Presently, aggregate is mixed with 5% cementitious material by weight. The cementitious material is composed of normal Portland cement, and ground blast furnace slag cement in the following ratios;

Number One Mine- 40 : 60, Normal Portland Cement : Slag cement
Number Two Mine- 67 : 33, Normal Portland Cement : Slag cement

The cements are slurried, at approximately 50% solids by weight, and are then sprayed onto the aggregate as it enters the stope or underground mixing station. All of this is outlined in the paper.

Do not hesitate to contact Dr. Yu or myself if you require further information, or other fill samples. Hoping to hear from you in the near future with your preliminary results, I remain,

Yours truly,

David B. Counter, P.Eng.
Associate Rock Mechanics Eng.

Kidd Creek Mines Ltd.
B-2002, Timmins, Ontario P4N 7K1
Mine Site (705) 267-2161

March 19, 1984

Mark Adams
Trow Ltd.
43 Baywood Road
Rexdale, Ontario
M9V 3Y8

Mark;

Please find enclosed a copy of our "design mix" grainsize chart. You should receive four pails of aggregate sometime this week. Three of the pails contain a scalped coarse aggregate,(-1.5 inches), while the fourth contains only fine aggregate,(-3/8 inch). They are clearly labeled.

From the curve you can see that the original design calls for twenty-five percent of the aggregate to pass a 3/8 inch screen. Our aggregate underground typically has twenty percent passing. You will have to juggle the numbers a bit to get the correct ratio of coarse to fines, but it should be in the range of thirty-five to forty percent passing 3/8 of an inch.

You should have enough material to cast nine 6" diameter cylinders. Typically, laboratory casts have had about twenty-three pounds of aggregate per cylinder. This means the aggregate has a bulk density of around 120 pounds per cubic foot.

Do not hesitate to call should you need anything else. If you require more aggregate, perhaps you could make a model mix from local ready-mix aggregates. If not, call and we'll send more. Hoping that this is satisfactory, I remain,

Yours truly

David B. Counter, P.Eng.

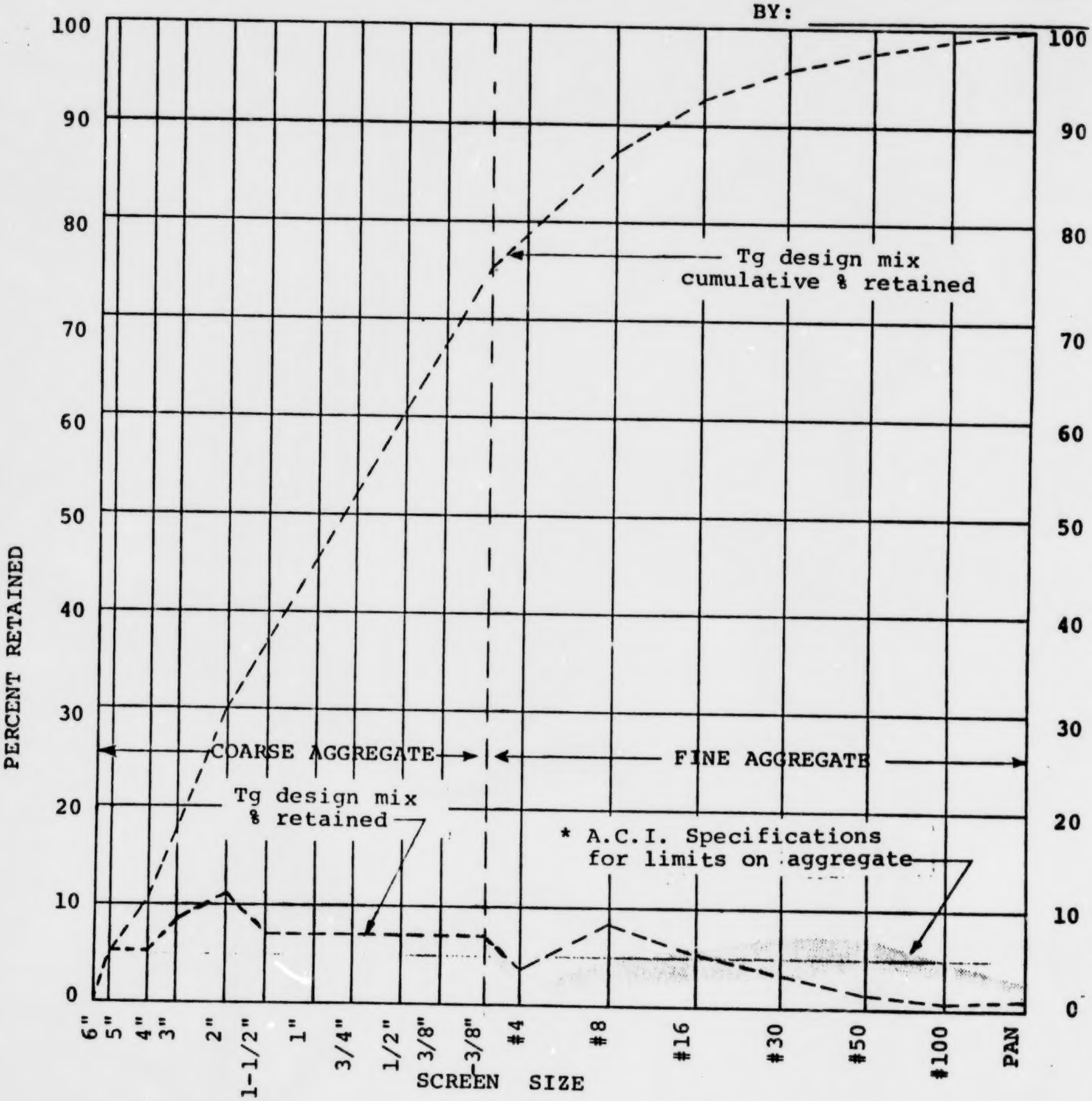
TEXASGULF CANADA LIMITED

B3.

BACKFILL AGGREGATE SCREEN ANALYSIS

DATE: ________________

BY: ________________



* American Concrete Institute

SAMPLE DATE	LOCATION	% COARSE	% FINES	REMARKS

Dist.: TY,DLP,EG,JPP,BF,CVA,JKY,JEB,BAT,TD

B4.

ROCK MECHANICS

Backfill practice and technology at Kidd Creek Mines

T.R. YU
Chief Engineer, Rock Mechanics
Backfill and Mine Research
D.B. COUNTER
Backfill Control Engineer
Kidd Creek Mines Ltd.

ABSTRACT

Kidd Creek Mines primarily employs sublevel blasthole stoping for underground mining operations. During pillar recovery in No. 1 Mine and panel mining in No. 2 Mine, the free standing fill exposures may reach a height of 100 m or more. Consequently, a competent consolidated rockfill has been required to fill the mined openings.

Since 1976, a total of more than fifty stopes and pillars have been filled with over four million tonnes of backfill. Fill exposures, up to heights of 100 m, have stood up well after blasting adjacent to the backfill.

The fill material consists of a graded aggregate, Portland cement and ground slag. Sand has been occasionally mixed in the fill material to serve as a void filler. The aggregate is prepared in a fill crushing plant, transported underground through raises, and hauled to the filling stopes by conveyors in No. 1 Mine and by trucks in No. 2 Mine.

Two mixing methods are currently in use. No. 1 Mine utilizes the free fall of gravity; the fill materials are sprayed with cement slurry as the aggregate departs from the conveyor into a mixing culvert. The other is the use of a rotary mixer on a batch basis in No. 2 Mine.

This paper describes the fill systems, filling operations, and the backfill quality control practice, together with various testing results related to improving fill quality as well as reducing fill costs.

Introduction

The Kidd Creek minesite, located 27 km north of the city centre of Timmins, Ontario, produces 4.5 million tonnes of copper, zinc, lead, and silver ores annually from its No. 1 and No. 2 underground mining operations. Both mines employ a sublevel blasthole stoping method.

No. 1 Mine was developed with a shaft to a depth of 930 m in 1973 to recover the ore between the floor of the open pit and a depth of 792 m. Stoping widths are normally 18 m, the heights vary from 90 m to 135 m, and lengths are from 22 m to 65 m. Vertical rib pillars between stopes are typically 24 m wide and sill pillars are in the order of 30 m thick.

Pillar recovery is implemented after filling the mined stopes with a consolidated rockfill. This allows a total recovery of the ore, with a minimum dilution, while also maintaining safe working conditions.

No. 2 Mine was developed from a second shaft, sunk to a depth of 1,530 m in 1978, for the recovery of ore beneath No. 1 Mine. Sublevel stoping in No. 2 Mine is similar to No. 1 Mine in method, i.e. blasting and filling, yet differs in mining sequencing, as no rib pillars are left.

Mining in No. 2 Mine began with a stope against the hanging wall, 15 m wide, 30 m long, and 60 m high, near the centre of the orebody. The mined stope was filled immediately with consolidated rockfill. After a three-week curing period, the adjacent stopes were subsequently mined by retreating toward the footwall, and expanding along strike on either side of the initial stope.

Currently, Kidd Creek Mines produces 4.5 million tonnes of ore annually. A total of 2.5 million tonnes of backfill are required to completely fill the mined out areas. Approximately

T.R. Yu

Thiann R. Yu, a 1957 mining engineering graduate of Cheng-Kung University, Taiwan, is chief engineer, Rock Mechanics, Backfill and Mine Research, for the Kidd Creek operations. His educational background includes an M.Eng. degree in mining (1967) and a Ph.D. in applied geophysics (1974), both from McGill University. During 1959 to 1965 he worked for the Taiwan Cement Company as a mining engineer. He was a research associate in the Department of Mineral Engineering at Ecole Polytechnique of Montreal before joining Kidd Creek in 1975. Dr. Yu is a member of the CIM, CNCRM, SEG, AGU and APEO.

D.B. Counter

David B. Counter, a native of Toronto, Ontario, graduated from the University of Toronto in 1981 with a Bachelor Degree in geotechnical engineering. Following his graduation, he joined Kidd Creek Mines Ltd. and is currently responsible for backfill quality control and research, as well as other geotechnical projects.

Mr. Counter is a member of the CIM.

Keywords: Rock mechanics, Backfill practice, Kidd Creek Mines Ltd., Blasting, Blasthole stoping, Underground mining, Aggregates, Portland cement, Ground slag.

80% of the fill placed has been consolidated material, consisting of cementing agents and aggregate crushed to a maximum size of 15 cm. Alluvial sand has been added in selected fills to increase the fine fraction, thereby acting as a void filler. The remaining 20% of the placed fill was unconsolidated rockfill or sandfill.

Before the present backfill system was established, study and research had been carried out for many years on the following items, among others:

1. Quantities of fill required annually for design of the plant;
2. Types of material available, e.g. alluvial sands, tailings, waste rock, etc.;
3. Fill strength requirements;
4. Method of fill preparation, mixing techniques, crushing facilities, screening, etc.; and
5. Material handling on surface and underground.

The result was the production of a strong consolidated fill formed by mixing aggregate crushed from pit waste with Portland cement as a binding agent, in the ratio of 20 parts aggregate to 1 part cement. This would ensure maximum ore recovery while minimizing dilution at a reasonable cost.

Preparation of Fill Materials

The decision to use consolidated rockfill at Kidd Creek necessitated the construction of both aggregate and cement handling facilities (Fig. 1). The value of the ore, the anticipated reduction in the amount of dilution, and the improved ground support provided by consolidated fill justified this added expense. It was also decided to incorporate as many of the existing pit ore crushing facilities into the future backfill sytem as possible, thereby reducing costs.

Cement Handling Facilities

Cement must be transported from surface in slurry form, as the size of the operation precludes the handling of bulk cement underground. On average, 500 tonnes of cementitious materials will be required daily to meet the annual projected fill requirement of 2.5 million tonnes.

Bulk cementitious materials are stored in three silos within the slurry plant. Two 500-tonne silos are used to store ground blast furnace slag, while the third 700-tonne silo is used to store normal type 10 Portland cement.

No cement production facilities exist in the Timmins area, and as a result, all cementitious products are supplied by either rail or truck from the Toronto area. Railcars are unloaded in a special unpressurized unloading facility at the minesite. Vibrators are attached to the railcars, and the cements flow by gravity, airslide, and pressurized pipeline to the top of the silos. Truck unloading is performed by "blowing" the cement under pressure into the silos from a separate unloading point.

The cements pass from the silos into a weighing hopper. The hopper has a capacity of 9 tonnes and the balance used to weigh the cement or slag is accurate to 10 kilograms. The slag is transported from the silos to the hopper by a 15-cm airslide, and the Portland cement is fed to the hopper by gravity through a 25-cm pipe. The hopper is capable of discharging the cements into either of two mixing tanks.

Each of the mixing tanks has a rated capacity of 11.9 m^3, with a six-blade impeller mixing the slurry. Water is fed into the mixing tanks by a 10-cm pipeline from either the recycled mine water ponds, or a fresh water lake. The water is measured by a flowmeter, and the tanks are equipped with high level indicators, electronic pulp density readouts and manual pulp density scales to assist in the correct mixing of slurry. Pulp densities can range from 15 to 70% solids by weight, depending on the fill design, but typical operating ranges are between 45 and 55% solids.

The plant operator controls which mine (No. 1 or No. 2) the slurry is sent to by remote actuation of valves. Once underground, the slurry destination is controlled by the underground workers. One tank is capable of sending slurry to both mines, while the other tank can send slurry to No. 1 Mine only. The system is capable of sequentially supplying slurry to both the north and south zones of No. 1 Mine, as well as both of the No. 2 Mine backfill stations. A batch takes approximately 15 minutes to mix and deliver to the underground fill location.

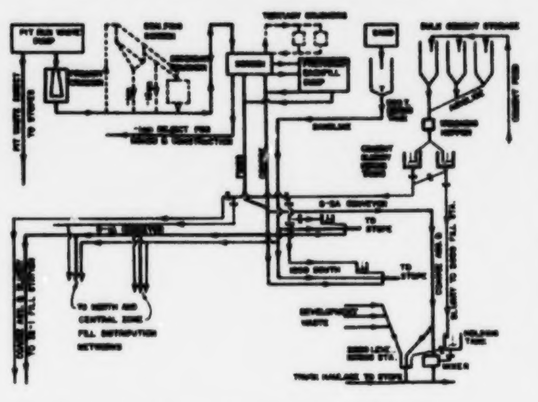


FIGURE 1. Kidd Creek backfill system—schematic.

Aggregate Production Facilities

The open-pit phase of the mine produced approximately 55 million tonnes of waste rock through a waste to ore stripping ratio of 2.5:1. This waste rock is now the primary source of fill material.

Waste rock is passed through the primary surface crusher (Allis-Chalmers 137 cm by 188 cm gyratory crusher, 14 to 15 cm O.S.S., capacity 1,340 tph) and directly to the backfill plant.

The magnitude of increased attrition on the aggregate with depth could not be foreseen during the initial design of the plant. Initial testing indicated that aggregate with a gradation of 75% -15 cm to + 1 cm, and 25% -1 cm by weight would, with 5% cement, provide a fill with a compressive strength of 7 MPa. Secondary and tertiary crushing of some of the aggregate provided the fines required when fill was being placed in the uppermost horizons of No. 1 Mine. With increasing depth, the attrition of the aggregate as it passed through the raises, made secondary and tertiary crushing unnecessary. Sufficient fines for No. 1 Mine are now produced by the primary crusher.

The secondary crusher previously used was a Telsmith 1700S, 122 cm gyrosphere crusher with a 15 cm O.S.S. and a rated capacity of 1200 tph. A scalping screen fed the crusher + 15 cm or + 8 cm aggregate as desired. The tertiary crushers, (Telsmith 48S, 240 tph, 3 cm O.S.S.) were used to produce more fines from a feed of -8 cm to + 1 cm aggregate.

Consequently, the backfill plant is capable of screening aggregate into four size ranges:

a) Coarse:	-15 cm to + 8 cm	
b) Midds:	- 8 cm to + 3 cm	75% by weight
c) Midds:	- 3 cm to + 1 cm	
d) Fines:	- 1 cm	25% by weight

The aggregates are blended and sent underground, via two silos, or outside where they are loaded into trucks and hauled to the prepared fill stockpiles. There is access to the two silos from outside the plant to avoid sending stockpiled material back through the system.

Some of the fines are rejected, as more fines are produced than are required for No. 1 Mine. They are used in construction and to make roadways. Fines sent from surface to No. 2 Mine form undesirable slimes which coat the coarser aggregate due to water entering the pass system.

The required amount of fines (-1 cm) and all of the coarse (-15 cm to + 1 cm) aggregates are sent underground separately into silos of 1500- and 4500-tonne capacities respectively.

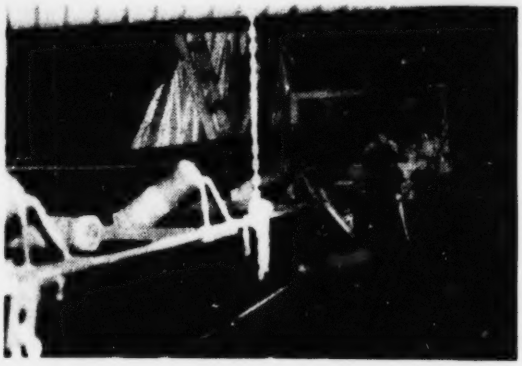


FIGURE 2. An underground backfill handling system.

Sand Plant

A new addition to the backfill system is the sand plant. Sand from a 150-tonne bin, is screened to -3 cm and fed into a 500-tonne mixing silo. A Marconajet mixing system slurries the alluvial sand in the bottom of the tank at a rate of 200 tph at a 65% pulp density.

The sand is sent underground via boreholes and 15 cm diameter A.B.S. plastic pipe. The pipe networks follow the conveyorways, in a manner similar to the cement slurry line.

Near the opening to be filled, the sand line and the 10-cm diameter pipe from the cement slurry pump join, and the mixed hydraulic fill is sent into the stope through a 200-mm borehole.

Underground Filling Systems

Aggregate Distribution

The aggregates pass via the silos from the backfill plant to the 8-2A sublevel, 180 metres beneath the surface. By simultaneous feeding of the coarse and fine aggregates onto the same conveyor, the aggregate can be mixed to the specified proportions of fine and coarse.

When aggregate is being sent to the No. 2 Mine fill stations, only the coarse feeder is used. Forano 137 cm by 305 cm, 10 h.p. imbalanced mercury weight, vibrating feeders are used throughout the system underground. They are capable of feeding up to 1200 tonnes of aggregate per hour, depending on the gate opening and conveyor speeds. The 8-2A conveyor is 0.9 m wide, 180 m long, and has a capacity of 1200 tonnes of aggregate per hour.

From 8-2A, the aggregate is sent to one of two raises. One raise carries coarse aggregate directly to the No. 2 Mine backfill mixing station on 2600 level (-792 m). The other raise is joined by two-finger raises to allow the aggregate to be distributed to three locations in No. 1 Mine.

One raise leads to the south orebody on 1200 level (-366 m) and a conveyor transports the aggregate at a rate of 600 tonnes per hour to the stopes below the 1200 level.

A second raise led to a temporary conveyor system in the south orebody on 800 level (-244 m). All stopes in this area have now been filled, and the system dismantled.

The remaining leg of the pass leads to a major distribution conveyor on the 8-1A sublevel, 15 metres above the northern end of 800 level. From here, coarse aggregate (fed specially from 8-2A) can be sent directly to the No. 2 Mine backfill mixing station on 32-1 sublevel (-945 m).

Mixed fine and coarse aggregate can be sent directly to stopes between the 800 and the 1200 horizons of No. 1 Mine, or it can be sent through a number of raises to all of the sublevels and main haulage levels in the northern zone.

Aggregate is also distributed to a hanging wall conveyor network by a conveyor through the main pit buttress pillar.

The 8-1A system is comprised of a 0.9 m wide belt, 360 m long, with a capacity of 1200 tonnes of aggregate per hour, and several splitters. All other conveyors operate at approximately 600 tonnes of aggregate per hour.

Three different forms of conveyor are used at Kidd Creek. The first form is used in "permanent" installations such as on 8-2A, and 8-1A sublevels. Steel rails support the idler and return rollers, and the rails are supported by steel legs firmly anchored to the drift floor.

The second form also uses steel rails as roller supports, but the rails are suspended from the back by chains. The use of these conveyors is more flexible as a prepared floor is not required for their installation. These conveyors are classed as "semi-permanent" and have a longer service life than the final form.

The third form of conveyor is truly portable, as the rollers are supported by wire ropes. The ropes are tensioned and supported by chains from the back (Fig. 2). The use of wire ropes as supporting members eliminates many of the problems of handling rigid conveyor sections underground.

Cement Slurry Distribution

All slurry is transported by gravity through 10- or 15-cm pipelines, and lined or unlined boreholes. In most cases, the slurry lines run along the conveyorways and parallel to the raises. In all cases, the slurry lines end at portable, skid mounted, holding tanks near the stope to be filled. The tanks, equipped with agitators, have capacities ranging from 5.7 to 11.9 m^3. In the case of the No. 2 Mine backfill mixing stations, these holding tanks are permanent installations.

Pipe networks are permanent installations, and are connected into the distribution system whenever a particular conveyor network is to be used.

Mixing of Aggregate and Slurry

No. 1 Mine

Many systems, such as mechanical agitators, slushers, sprays, vibrating mixing conveyors, etc. were considered during the design of the backfill system.

The simplest system found to date employs a baffled culvert or chute, a spray header for the slurry, which is pumped from the holding tank, and the conveyor carrying the aggregate.

The conveyor discharges the aggregate into a 1.2-m diameter steel culvert, 2 to 3 m long, which is equipped with two or three baffles set at 55 degrees to the axis of the culvert. The slurry is sprayed on the aggregate as it enters the culvert, and the tumbling action of the aggregate as it passes from baffle to baffle ensures that each aggregate receives a good coating of slurry. The mixed fill then falls through a 0.7-m diameter bored, downreamed raise, into the opening being filled. The dip of the 0.7 m raise is typically greater than 55 degrees.

When an opening is directly accessible from a fill level, a chute, 1 m wide by 4 m long, replaces the culvert. The use of the slurry header and baffles is the same as in the culvert. The chute can be advanced with the conveyor out onto the fresh fill to tight fill the stope.

Mixed fill can also be hauled by teletram or scooptram when the tonnage of fill required to fill a location does not justify the expense of installing a conveyor.

No. 2 Mine

Because both the 32-1 and 2600 mixing stations are similar, only a basic overview of items typical to a station will be provided.

From the raise, the aggregate passes into a 4.5 m^3 metering pocket equipped with two guillotine gates. The aggregate is then dumped from the pocket into a pivoting tipple chute, capable of feeding either of two 5 m^3 redi-mix concrete mixer drums. Mounted discharge end to discharge end, the mixers operate singly or in tandem. Slurry is pumped to a measuring hopper equipped with an overflow outlet. When the hopper is full, small quantities of slurry will overflow into either mixer, signalling the station operator to shut off the slurry pump. The full "shot" of slurry is fed by gravity through a 10-cm diameter pipe, lying within the tipple chute, into the mixer. The slurry and aggregate are then mixed until all of the aggregate is coated. This entire cycle takes approximately two minutes.

Through the use of air pistons, the entire mixer and its drive pivot around a horizontal line, 90 degrees to the mixer axis, to allow dumping of the mixed fill into underground haulage trucks. While either mixer is discharging, the other can be filled. It is also possible to operate the mixers independently to allow time for maintenance on one without disrupting the operation of the other.

Low-profile trucks (Jarvis Clark, 9- and 11.8-tonne) haul the fill to the stope, where it is dumped from a drill drift directly into the void. A concrete bumper block prevents the trucks from falling into the stope. An average circuit time from the mixing station to the dump point and back is about 7 minutes. As it takes about 1 minute to discharge the mixer, the total cycle time is around 8 minutes, and 4 trucks operate very efficiently with each station.

Each station can prepare approximately 3000 tonnes of fill per day. The stations are also equipped with an aggregate bypass finger raise and slurry discharge pipe to allow unmixed fill or development waste to be placed in the event that both mixers are inoperable. The slurry is sprayed over the muck, and all coating takes place during dumping and impact.

Backfill Design and Quality Control

With the established requirement of 7 MPa backfill, some means of ensuring its production had to be devised. Also, specific stopes require other fill properties to be enhanced, for example:

1) increase or decrease the angle of repose of the fill to form a required mining line:

2) decrease the curing time required to allow early pillar recovery, yet maintain the strength characteristics; and

3) allow the fill to remain unsupported over a span of 20 or 30 metres or more, etc.

For this reason, the fill is designed with fairly strict accordance to established criteria, yet the design is flexible enough to meet production requirements without sacrificing quality.

Design

The original backfill design has been maintained since its inception six years ago. Minor changes have been made to allow greater flexibility in the mining method.

No. 1 Mine

Once an opening has been designed, and a mining sequence decided, the backfill design can be formulated. Design variables include:

1) the fill type, i.e. consolidated, unconsolidated, etc.;

2) anticipated free standing height, or span of fill, during pillar recovery;

3) cement: aggregate ratios (if required);

4) cement type and proportioning, Portland: slag (if required);

5) the belt speed, or haulage equipement and the raises required; and

6) the need for retarder.

From the beam and arch theory, the cement content for self-arching fill can be determined, together with the depth of the cement-enriched sill required, based on the anticipated undercut exposure dimensions (Fig. 3).

With the cement contents of the sill and the remainder of the stope, normally 5%, now established, the water requirements and slurry batch sizes may be determined for a given conveyor speed. Experience has shown that aggregates and sand require moisture contents of 3% and 30% respectively to flow. The water content of the slurry may be altered for special angles of repose, etc. Limiting factors include the size of the slurry mixing tank on surface and the rate at which slurry can be delivered underground.

Before filling begins in each stope, a proposal is issued for comment, and if approved, operating procedures are issued to the production department. These indicate the belt speed, aggregate tonnage, tonnages of cementitious materials and water per batch, plus any other special procedures to be followed.

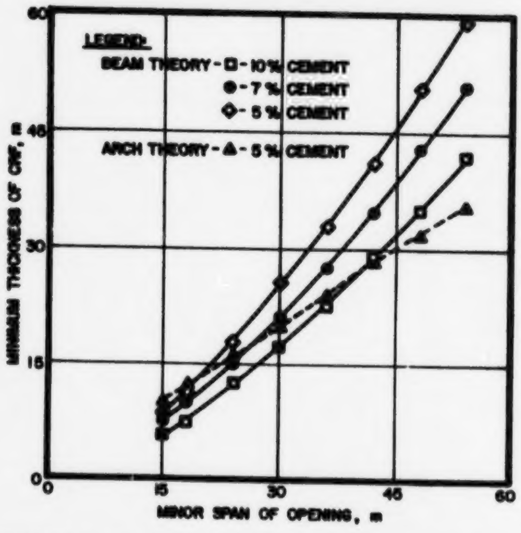


FIGURE 3. Minimum thickness of CRF vs span of opening.

No. 2 Mine

Due to the nature of the backfill mixing station and mining procedures, little redesign has been done for No. 2 Mine since the beginning of operations about two years ago.

The cement requirements were established at 4.2% and the automated plants were set up to operate on a batch basis. To vary the cement content in a stope, only the number of trucks of aggregate per batch of slurry needs to be altered.

Retarder

Retarder is required now only in areas where pure Portland cement is used as a binder, as the slow curing rate of the slag naturally delays the final set of the fill. The purpose of the addition of a retarder will be described later.

Quality Control

Maintaining the quality of the backfill at Kidd Creek is of prime importance.

Observation of the fill placed in stopes provided much of the initial data for quality control. The water requirements were determined in this way, as it was easy to observe which mixes flowed well, which had too much water, resulting in gullies and rills, and which had too little water, resulting in a dry, unworkable mix.

Daily inspection of filling operations by supervisory and engineering staff maintains this high level of quality. Daily records are kept of the tonnages of all fill materials placed underground, and this allows trends to be noticed, and problems corrected before they become major. For example, low cement contents in No. 2 Mine fill could be an indication of excessive scale building up in the slurry measuring hopper, etc.

Periodic screening of aggregates indicates whether or not fewer fines should be added to the coarse aggregate on 8-2A sublevel, or if the primary crusher should be reset to maintain the desired gradation.

Slag is routinely tested for compressive strength to meet CSA Standard A363 at the production facility near Hamilton before it is shipped to the mine. Similar testing, done at the minesite to confirm these results, guarantees the quality of this material.

Types of Consolidated Fill

Since the beginning of the filling operation in 1976, various types of backfill have been developed in the mines to meet the varied individual stope requirements. Fill materials now include

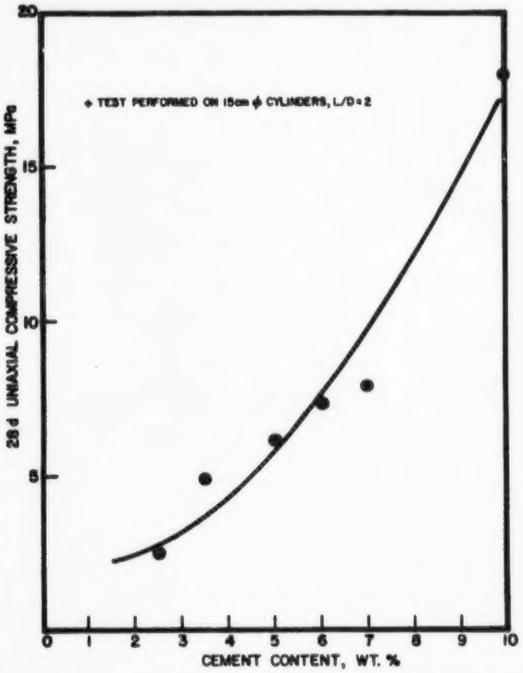


FIGURE 4. Optimum strength of CRF⁺ vs cement content.

crushed aggregates and alluvial sand, with Portland cement and ground blast furnace slag as a blended binding agent. Retarder was also used in the cement slurry for some time.

Consolidated Rockfill (CRF)

The original consolidated rockfill(1) was a graded aggregated with Portland cement in the ratio of 20:1. This design was calculated from the strength requirements which were necessary for the fill to stand over an exposed face 120 m high or more, by approximately 70 m long.

Considering gravity loading alone, the fill would require a compressive strength of 2.8 MPa after 28-day curing. A safety factor of 2.5 was applied to allow for additional blast loading and the reduction of fill strength due to inadequate mixing, thus arriving at a design strength of 7 MPa.

Testwork revealed that graded aggregate containing 75% coarse material (-15 cm to 1 cm) and 25% fines (-1 cm to No. 100 mesh) would satisfy both material handling and strength requirements.

Figure 4 shows the relationship between the optimum strength and the cement content for 15 cm ϕ CRF cylinders. It is to be noted that great care had to be exercised in the laboratory tests to obtain the optimum strength at a given cement content. Such optimum conditions, however, might not be reproducible underground, and the over-all fill strength in the stopes, therefore, would be expected to be somewhat lower than those shown in Figure 4.

To simulate the actual stope fill operation, several large-scale dump tests were carried out before filling began underground. Table 1 lists some of the measured physical properties of core specimens or samples taken from the dump tests, together with results from tests on placed fill. It can be seen that each test revealed an adequate strength of over 6 MPa, with a peak strength of 10.3 MPa in core specimens from 838 stope. However, it was found from chemical analysis that the cores contained a high cement content, 7.3% as compared to the average of 5%. This indicated that cement concentrations could occur in some areas of large fill piles, resulting in undesirable deficiencies of cement in other areas.

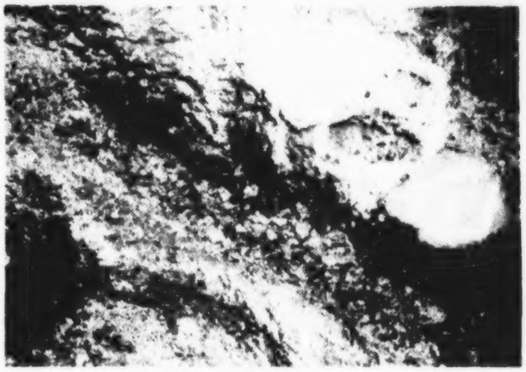


FIGURE 5. Layering of CRF in 1235 stope.

In 1978, a test drift 14 m long was driven in a stope which had been filled one year earlier with consolidated rockfill. The drift was excavated to examine the operation of a continuous mucking machine in CRF, and to evaluate the support requirements of excavations in the fill material. The drift also enabled various in-situ strength tests to be performed.

To date, the drift shows no sign of deterioration, even though the adjacent pillar, 10 m away from the drift, was mined out in 1979.

Observation of the drift gives further insight into the distribution of CRF in a stope. As shown in Figure 5, the fill exhibits alternating layers of coarse and fine aggregate resulting from slumping and uneven aggregate distribution during placement.

From a series of plate-load test and laboratory strength tests, the in-situ strength of the fill material in the test drift was determined, as shown in Figure 6. It can be seen that the strongest fill, with an average strength of 11 MPa, was obtained within the dump cone area. The strength decreases as the aggregate in the layers gets coarser. From the distribution characteristics of the fill and the relationship between the strength and the cement content, it is clear that a zone of a high cement content tends to occur near the dump cone area, leaving a lean cement rockfill at the perimeter of the fill cone. Such segregation phenomena should be eliminated because a strong fill near the stope perimeter is desirable.

Since 1976, a total of more than fifty stopes and pillars have been filled using over four million tonnes of consolidated rockfill. Fill exposures, up to heights of 100 m, stood well even after blasting adjacent to the fill as shown in Figure 7. Some minor failures which have occurred can be attributed to segregation effects.

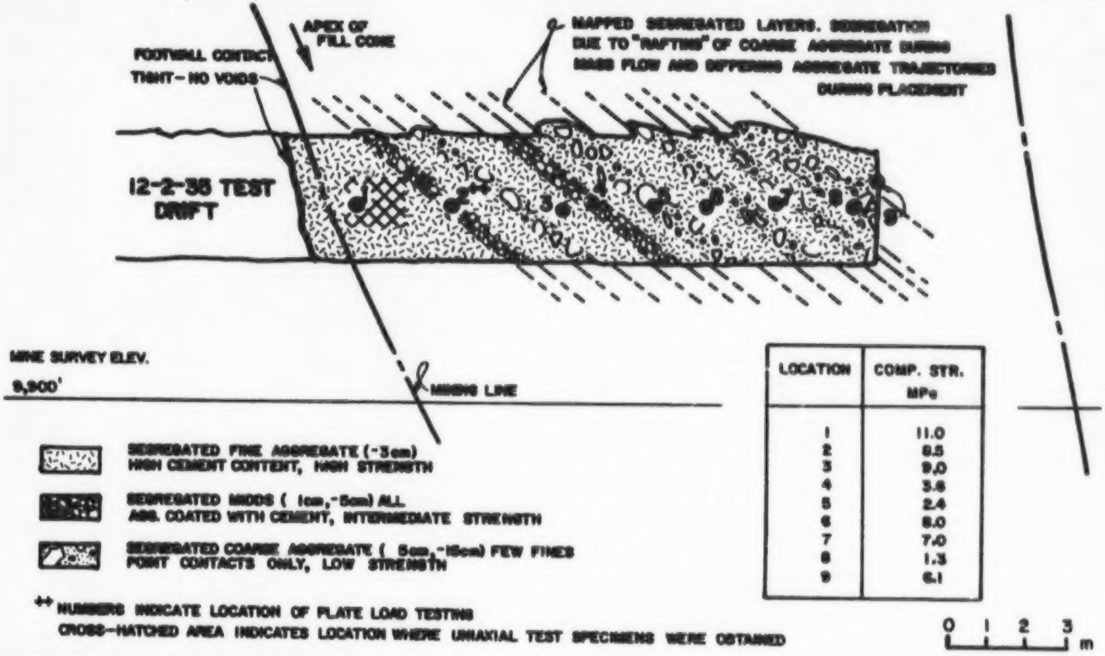
Consolidated Sandfill (CSF)

This type of fill is essentially a cemented hydraulic fill using alluvial sand in place of normal tailings. The sand, from an esker on the mine property, has 90-95% by-weight passing a No. 18 mesh (1.0 mm) screen, and has an average percolation rate of 15 cm/hr. This sand, when mixed with cement slurry, becomes an ideal material to fill the voids in segregated aggregate or waste. Figure 8 illustrates typical test results of strength vs cement content for 7.5-cm diameter CSF cylinders.

Three uses of cement sand slurry have been identified and are listed below:

CSF for Waste Consolidation

The cement sand slurry, with a richer amount of cementing agent, is introduced into mined stopes to consolidate waste in place. Normally, the cement content in the sand slurry is adjusted so that the over-all cement in the final product is about 4.8%. The amount of sand required for filling 100% of the voids in typical waste is about 30% by-weight of waste.


APEX OF FILL CONE
FOOTWALL CONTACT
TIGHT - NO VOIDS
MAPPED SEGREGATED LAYERS. SEGREGATION DUE TO "RAFTING" OF COARSE AGGREGATE DURING MASS FLOW AND DIFFERING AGGREGATE TRAJECTORIES DURING PLACEMENT
12-2-35 TEST DRIFT
3
MINE SURVEY ELEV.
9,900'
℄ MINING LINE
SEGREGATED FINE AGGREGATE (-3cm) HIGH CEMENT CONTENT, HIGH STRENGTH
SEGREGATED MIDDS (1cm,-5cm) ALL AGG. COATED WITH CEMENT, INTERMEDIATE STRENGTH
SEGREGATED COARSE AGGREGATE (5cm,-15cm) FEW FINES POINT CONTACTS ONLY, LOW STRENGTH
↔ NUMBERS INDICATE LOCATION OF PLATE LOAD TESTING
CROSS-HATCHED AREA INDICATES LOCATION WHERE UNIAXIAL TEST SPECIMENS WERE OBTAINED
0 1 2 3 m

LOCATION	COMP. STR. MPa
1	11.0
2	8.5
3	9.0
4	3.6
5	2.4
6	8.0
7	7.0
8	1.3
9	6.1

FIGURE 6. Variation of fill quality with location in 12-2-35 test drift—1235 stope, section looking north.

FIGURE 7. A view of three different fill exposures from 12-2-25XC.

CSF and CRF

With the same cement content as CRF, the cement sand slurry is introduced simultaneously with the CRF to fill the voids in segregated aggregate as they form. The amount of sand required for this purpose is about 15% by-weight of CRF.

CSF as Bulkfill

With a lean cement content, the CSF is placed after the majority of the stope has been filled with CRF to tightfill the remaining void beneath the stopeback, thus providing ground support.

Consolidated Sand Rockfill (CSRF)

As evidenced in the test drifts in 1235 stope, significant segregation of coarse aggregate took place in the outer zone of the fill pile. The segregation would also occur in the areas adjacent to stope walls even within a small "rolling distance" from the dump points. Such common phenomena caused weak zones in the underground fill prone to blasting damage during pillar extraction.

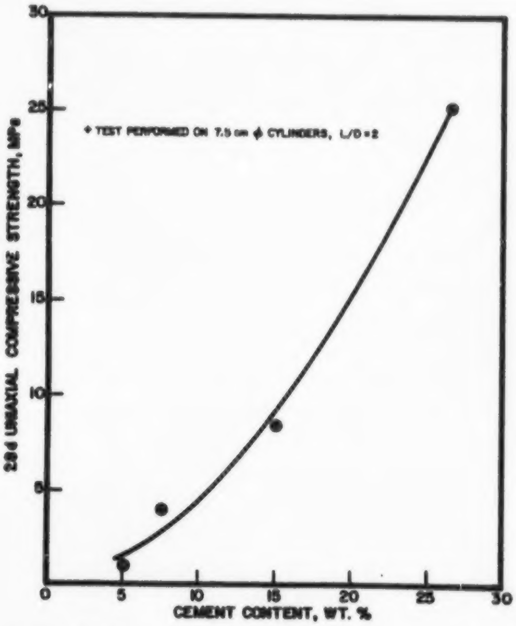


FIGURE 8. Strength of CSF⁺ vs cement content.

To minimize the effect of segregation without increasing the number of fill dump points, tests were conducted by adding sand into the CRF to act as a void filler.

Figure 9 illustrates the relationship between sand content in the CSRF and the compressive strength. The increased contact area between aggregate particles at low sand contents, say at 5%, has resulted in a 40% increase in compressive strength. When more sand is added, the increase in the surface area of the aggregate, which must be coated with the same amount of cement paste, causes the strength to decrease monotonically.

The test results imply that sand addition to the CRF should be kept at 5% for an optimum strength. This design mix, however, is not feasible in practice, because sand slurry tends to fill every void completely.

Under a "sand saturated" condition, i.e. 100% of the voids filled, the compressive strength of CSRF is only one third of the strength of CRF with the same cement content, as shown in Figure 9. If the required fill strength was solely a function of the compressive strength, then the addition of sand would not be beneficial. However, if the ability of fill to resist blasting vibration damage is governed by the dynamic tensile strength, the sand addition will be justified.

To verify this concept impact tests were conducted, with a large Schmidt hammer acting as a constant source of dynamic loading, simulating the blast vibration. It can be seen from Figure 10 that the most impact resistant fill was obtained when the aggregate had been fully surrounded by a sand-cement matrix, thereby enhancing the over-all stability of the structure.

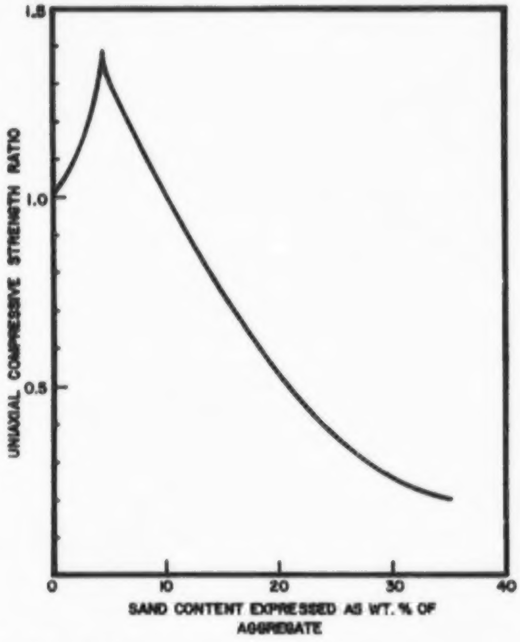


FIGURE 9. Compressive strength vs sand content.

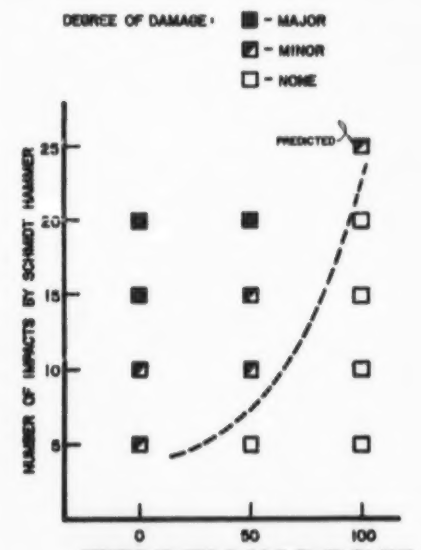


FIGURE 10. Impact resistance testing with Schmidt hammer.

Another reason for the use of CSRF is that the need for coating all aggregate with cement slurry becomes less critical. This implies that a lesser degree of mixing of cement slurry and aggregate is acceptable and the number of fill raises required, therefore, can be more flexible.

Based on the laboratory test results and the evaluation of three stopes filled with CSRF, the construction of a sand slurry plant was justified. To date, the sand plant has been constructed and is operational.

However, due to the extra cost of the sand slurry, the use of CSRF is limited to selective stopes where a more stable fill exposure is required.

Cement Slag Rockfill (CSLRF)

This type of fill refers to consolidated rockfill incorporating a blended binder of Portland cement and ground blast furnace slag. The replacement of a portion of the Portland cement with ground slag yields a moderate cost reduction.

Figure 11 illustrates the laboratory test results of cement slag rockfill cylinders. It can be seen that at 85 days the strength of CSLRF with 2.5% cement and 2.5% slag by-weight as a blended binder is equivalent to the strength of a 5% cement mix. The addition of 5% sand enhances the strength of CSLRF by 45%.

Figure 12 depicts the relationship between the compressive strength and the ratio of ground slag to Portland cement at various ages. The graph has been used as an aid in designing a proper blended mix in a stope for a given allowable curing period.

To evaluate the curing process of cement slag rockfill in an underground environment, fourteen large test blocks (60 cm by 60 cm by 100 cm high) were cast underground (Fig. 13). The dynamic strength characteristics were determined by employing the impact method with a Schmidt hammer as mentioned earlier, and by the detonation of blasting caps within the

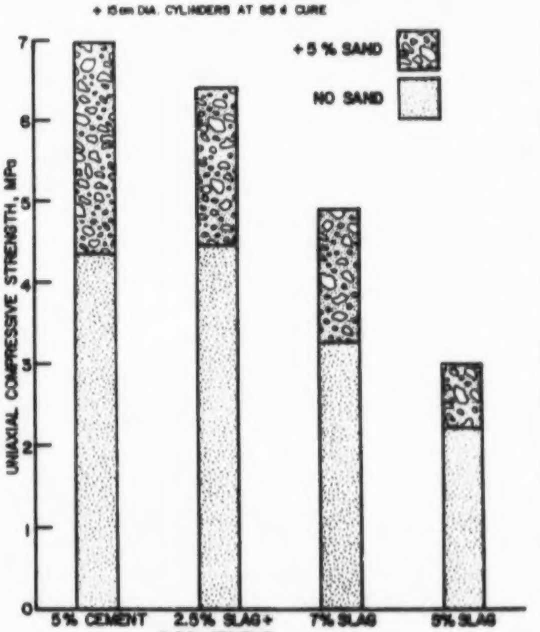


FIGURE 11. Effect of cement/slag/sand content on the static strength of backfill⁺.

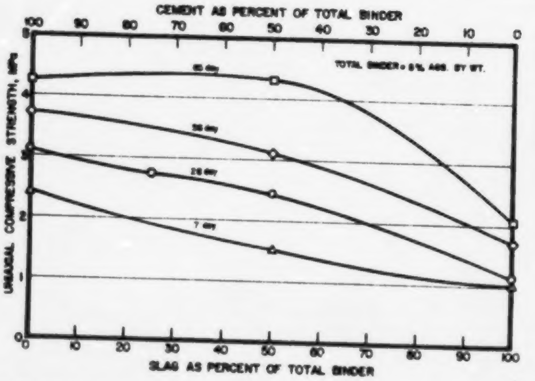


FIGURE 12. Uniaxial compressive strength vs slag/cement ratio.

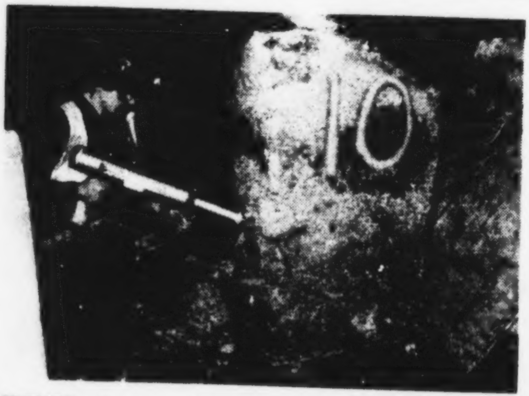


FIGURE 13. Schmidt hammer test on CSLRF blocks underground.

blocks. This allows the evaluation of the degree of blasting resistance by the relative size of the craters.

The test result confirmed the previous observation that blocks containing 15% sand possess a higher dynamic resistance than those with a lower sand content. After more than two years of testing, the following conclusions have been made:

1) Ground blast furnace slag can replace up to 50% of the Portland cement in consolidated rockfill without loss of its compressive strength provided that the fill is allowed three months to cure.

2) The mix of Portland cement and slag with sand slurry also produces an acceptable fill, but requires longer curing periods to obtain the desired strength.

3) No operational problems were encountered in the use of blended cement and slag fill in a waste pass. This large scale test also revealed that the slow curing nature of slag eliminates the need of a retarding admixture, resulting in a further cost saving.

Since 1982, Kidd Creek has been using ground blast furnace slag as a partial replacement of Portland cement in the backfill. In No. 1 Mine, wherever a minimum three-month curing period is available before adjacent pillar recovery, up to 60% of the Portland cement has been replaced by an equivalent weight of ground slag with satisfactory results. Due to a shorter curing period available in the No. 2 Mine, normally three weeks, only 33% replacement of Portland cement by ground slag is allowable.

By the end of February 1983, a total of 28,000 tonnes of ground slag was used in the backfill. Several exposures of CSLRF have verified the laboratory test results, and there have not been any significant fill failures which can be attributed to the replacement of Portand cement by ground slag.

Fill Research and Development

At Kidd Creek Mines, producing a quality backfill has been recognized as an essential step toward more successful pillar recovery. Progress has been made to improve the fill quality by modifying operating procedures and through various laboratory tests. Nevertheless, future study and testwork will further upgrade the fill product, with the ultimate objective being stronger, more economic backfill.

Some of the testwork which has been carried out at the minesite is summarized as follows:

Mixing of Cement Slurry and Aggregate

An important factor in producing a strong consolidated fill is to have all of the aggregate coated with the supplied amount of cement slurry in a very short time. Prior to, and during start-up of the fill operations, an intensive study had been made on this subject, including full scale testing of a vibratory mixing conveyor, the use of a slusher for mixing, a baffled mixing culvert, a drum mixer, etc.

Presently, baffled mixing culverts are used exclusively in No. 1 Mine and drum mixers are employed in No. 2 Mine.

Impact Damage to Partially Cured CRF

The impact of aggregate falling from the fill raise onto placed CRF, which is already beyond the final set time of the cementing agent, causes permanent broken bonds. This problem is easily overlooked in smaller operations, where fill is either placed hydraulically, or at such a slow rate that the damage is not noticed.

In an attempt to reduce the extent of impact damage, a retarding admixture was added to the cement in the past. The retarder created a buffer layer of CRF, a minimum 60-cm thick, in which final set had not occurred. This allowed the shock of freshly dumped fill to be absorbed by the plastic surface layer, preventing impact damage to the cured fill beneath.

Some tests on the use of ligno sulphonate as a retarder substitute were made. A defoaming agent was necessary, however, as an agitated lignosol-cement slurry mix foams quite readily, resulting in pumping problems.

Another advantage of adding a retarding admixture to the cement slurry was a reduction in the rate of cement scale build-up in the slurry pipelines. The admixture is not required if a blended cement slag binder is used, because the blended binder cures at a much slower rate.

Static and Dynamic Backfill Strength Characteristics

When a pillar adjacent to a filled stope is blasted, the fill must be able to sustain the gravity loading of the overlying fill material, as well as the dynamic load applied during blasting.

It is well known that the most important parameter governing the stability of gravity loaded fill is the cohesion of the material. This is commonly referred to as the static strength of the fill.

The parameters governing the increased resistance to blast damage against CRF, however, are not fully known yet. It is postulated that the dynamic tensile strength may be closely related to the blasting vibration resistance.

Some materials such as consolidated rockfill or concrete are strong in compression, yet weak in tension. The dynamic tensile strength of CRF, therefore, should be improved if the degree of blast damage is to be reduced.

Laboratory tests have revealed that the addition of sand to the CRF, as mentioned earlier, has not only reduced the effect of segregation, but also has enhanced the dynamic strength, and thus the resistance of the fill to blast damage.

In-Situ Strength Determination—Plate-Load Test

A series of compressive strength and plate-load tests were con-

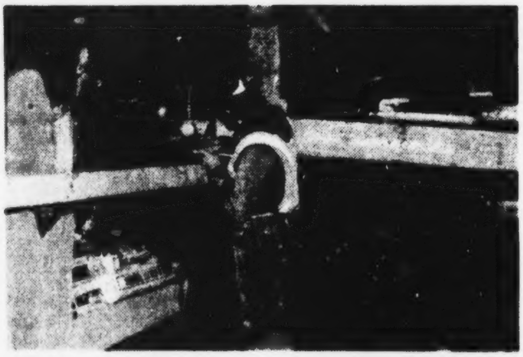


FIGURE 14. In-situ strength test of CRF in 12-2-35 test drift.

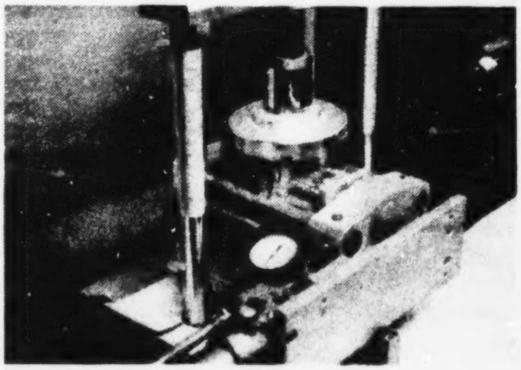


FIGURE 15. Laboratory direct shear test.

ducted on backfill material from the test drift in 1235 stope (Fig. 14), to develop a quantitative method of determining the in-situ strength of the fill product.

Conclusions drawn from testing indicated that the bearing plate-load test would provide a quick and reliable method of determining the in-situ strength of CRF. The ratio of ultimate bearing strength to uniaxial compressive strength for CRF was found to be approximately 3.2 to 1.

Laboratory Direct Shear Test

In analyzing fill stability, the shear strength is an important variable. The shear strength of CRF was determined using the direct shear test as shown in Figure 15. The results, plotted in Figure 16, show the relationship between the shear strength and the normal stress. As measured from the graph, the apparent cohesion and the apparent friction angle are 1.1 MPa and 33 degrees respectively. This corresponds well with the observed angle of repose of the fill, normally 34 to 37 degrees.

Test of Portland Cement Substitutes—Fly Ash, Copper Slag and Blast Furnace Slag

Considerable savings could be realized by minimizing the amount of Portland cement used with partial replacement of materials which are more cost efficient.

The partial replacement of Portland cement with power station fly ash was tested in the lab with an acceptable strength result. The longer curing times and the inconsistent chemical compositions, however, impeded its use. Nevertheless, the potential uses of fly ash will be investigated further.

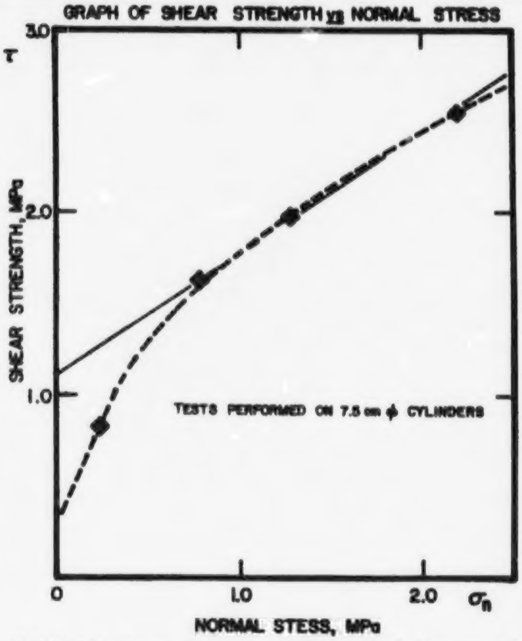


FIGURE 16. Direct shear test on CRF graph of shear strength vs normal stress.

TABLE 1. Some physical properties of CRF from drop tests and underground fill specimens

Sample	Density g/cc	Curing Period	Ave. Comp. Strength MPa	Elastic Modulus GPa	Remarks
Three cores 15 cm dia.	2.0	28 days	6.9	2.0	Samples from a 110 m drop test.
Three cylinders 15 cm dia.	2.5	28 days	6.1	-	-3.8 cm fill screened from a 24 m drop test.
Nine cores 8.4 cm dia. from 838 stope	2.3	3 mths	10.3	3.1	Cement content 7.3%.
1235 stope; Dump Cone	2.4	4 yrs	11.0	3.8	Nine samples tested in lab.
Fines Layer	-	*	8.5	-	In-situ plate load test.
Mid Agg. Layer	-	*	3.5	-	In-situ plate load test.
Coarse Agg. Layer	-	*	Est. 1.3	-	-

Evaluation of the pozzolanic properties of ground copper slag have been going on for several years. Initial test results indicated that the compressive strength of a sand mixture with 2.5% cement and 9% ground slag obtained a strength equivalent to that of the same sand mixture with 5% cement, after a curing period of 112 days. Nevertheless, the ground copper slag has not been used owing to the slow curing process and the cost of constructing grinding facilities at the present time.

After intensive testwork on ground blast furnace slag prior to January 1982, the mines started to blend Portland cement with ground slag as a cementing agent for backfill, with the mix proportions as mentioned earlier.

Conclusions

Based on the experience of six years of fill operations, and on various laboratory tests conducted over the years, the following conclusions have been made:

The backfill system developed at Kidd Creek Mines is unique in terms of its flexibility as a material handling system, the short lead time required for fill preparation and the reliability of the operation.

The use of consolidated rockfill in place of typical cemented hydraulic fills was the correct choice for this operation. This has been reflected in the excellent performance of the backfill as an aid in ground support, in the reduced dilution of muck, and the full recovery of ore.

The quality of the fill is also a reflection of the teamwork between the fill operators, the production supervisory staff, the engineering staff, and the management.

Consistent efforts to improve the fill quality, while searching for a means of reducing costs, have resulted in some beneficial results, such as the use of sand as a void filler to increase the fill stability, the use of ground slag as a partial replacement of Portland cement to generate a moderate cost advantage, etc.

At Kidd Creek, the important role that backfill plays in mining has been recognized to such an extent that the fill proposal is an integral part of the over-all mining plan.

It is believed, however, that a continuous R & D program on backfill will be able to improve the fill quality and to reduce the operating costs even further.

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